MIR PUBLISHERS

Акад. А. А. СКОЧИНСКИЙ, проф. В. Б. КОМАРОВ

Рудничная вентиляция

ИЗДАТЕЛЬСТВО «НЕДРА» • МОСКВА

A. SKOCHINSKY and V. KOMAROV

MINE VENTILATION

Translated from the Russian

by

JOHN S. SCOTT

Translation Edited

by

DMITRI A. TELYAKOVSKY

MIR PUBLISHERS * Moscow 1969

UDC 622.42/.46(075.8)–20

TO THE READER

Mir Publishers would be grateful for your comments on the content, translation and design of this book. We would also be pleased to receive any other suggestions you may wish to make.

Our address is: Mir Publishers, 2 Pervy Rizhsky Pereulok, Moscow, U.S.S.R.

Printed in the Union of Soviet Socialist Republics

На английском языке

CONTENTS

PART ONE

THE MINE ATMOSPHERE

PART THREE

PLANNING OF MINE VENTILATION

PART ONE

The mine atmosphere

CHAPTER 1

MINE AIR

1-1. MINE AIR AND ITS VARIETIES

Mine air is a mixture of vapours and gases, almost always dusty, which fill the underground workings. It is atmospheric air which comes underground and suffers various alterations in its composition. If the changes are so small that the mine air can be regarded as atmospheric air, it is described as *fresh* or *intake* air, stale or vitiated air being described as *exhaust* or *return* air.

Atmospheric air is a mixture of nitrogen*, oxygen, carbon dioxide, and water vapour, the first three components being in the constant proportions of about 79%, 20.96%, and 0.04% (by volume), respectively. The water vapour contained in air varies around an average 1% by volume, which does not affect the $N_2 : O_2$ ratio**.

Atmospheric air entering the mine undergoes changes in its composition.

Generally speaking, these changes involve a *reduction* in the amount of *oxygen*, and an *increase* in the content of *carbon dioxide* as well as addition of the following impurities:

(1) *Harmful gases.* These are suffocating, poisonous or explosive gases (methane, nitrogen, carbon monoxide, hydrogen sulphide, hydrogen, etc.) or, in mines which work uranium and thorium deposits, gaseous radioactive substances (radon, etc.).

(2) *Harmful vapours* (mercuric, arsenic, etc.).

(3) *Dusts* and *fumes.*

The temperature, pressure, specific gravity, and moisture content of air vary in both directions

The degree of contamination of the mine air mainly depends on the following factors:

---

* Atmospheric nitrogen is usually understood to include the inert, so-called rare gases: argon, krypton, helium, xenon, and neon; there is no practical need to separate them from nitrogen, so far as mine ventilation is concerned.

We should note, however, that the ratio of the content of the rare gases to the content of nitrogen in the gases of the earth's crust, may indicate their origin If the ratio is close to that characteristic of atmospheric air (1 18-1.20 per cent), this would be an indication of their atmospheric origin

** Some investigators of the mine air take the gaseous content of normal atmospheric air to be as follows 79.04% N_2, 20 93% O_2, 0.03% CO_2 by volume

(1) The gas content of the minerals and rocks being mined.
(2) The quantity of air moving through mine workings.
(3) The length of the workings
(4) The tendency of minerals and rocks to absorb oxygen and to oxidize
(5) The method of working.

All other conditions being equal, the air in mine workings becomes more contaminated (a) in mines or in mine districts where the air flows slowly, and (b) when the mine airways are only slightly branched and the air flows through a long circuit.

Thus, even slightly fouled mine air always contains O_2, CO_2, and N_2 in the amounts differing from those in pure atmospheric air. Therefore we can regard mine air as consisting of three parts: atmospheric air, "active gases", and dead air. By *active gases* are meant any poisonous or explosive gases which are liberated or formed underground and mix with the mine air. *Dead air* is a mixture of CO_2 and N_2, contained in the mine air in a quantity exceeding their content in atmospheric air. Dead air is devoid of oxygen. As a component of the mine air, dead air lowers the oxygen content

The composition of dead air in different mines varies within wide limits: 5-15% (sometimes more) CO_2 and 95-85% N_2. The so-called index of dead air, $K = \frac{CO_2}{N_2}$, also varies in wide limits. But for a particular mine (especially for any district) this index is constant; its variation in either direction is an indication of a disturbance of the normal gas exchange in the mine (or mine section)

Dead air should not be regarded as something fictitious; its content in the air of the mine workings being ventilated usually varies from a fraction of one per cent up to 20%, but in old (sealed) workings its content may increase up to 80% or 90% and even higher.

There are well-known instances of emission of 100% dead air. For example, in a Californian gold mine there was an emission of dead air in such a quantity that some of the miners were suffocated, this air containing 13 9% CO_2 and 86.1% N_2 came from a carbonaceous layer in an igneous rock.

The examples below show how to calculate the content of dead air in the mine atmosphere and how to determine its composition from the data of laboratory analysis of samples of the mine air.

Example 1. The general return air current of a coal mine contains. 0 650% CH_4 + 0 305% CO_2 + 20 340% O_2 + 78 705% N_2 = 100%

The composition of the atmospheric air entering the mine is 0 03% CO_2 + + 20 93% O_2 + 79 04% N_2 = 100% The nitrogen equivalent of 20 34 litres of oxygen (i e , the volume of N_2 corresponding to the above volume of O_2 in

pure air) is:

$$N_2 = 79.04 \times \frac{20.34}{20.93} = 76.81 \text{ litres}$$

The carbon dioxide equivalent of 20 34 litres of oxygen is

$$CO_2 = 0\ 03 \times \frac{20.34}{20\ 93} \approx 0\ 03 \text{ litre}$$

Consequently, the differential composition of the return air is:

Atmospheric air		
CO_2 0.03%, O_2 20 34%, N_2 76 81%	. . . .	97.18%
Active gases (methane)	. . . . .	0.65%
Dead air		
N_2 = 78 705 − 76 81 = 1 895; CO_2 = 0 305 − 0 03 = 0 275	. .	2 17%
Total	. . . . .	100 00%

The composition of the dead air is

$$N_2 = \frac{1.895}{2\ 17} \times 100 = 87\ 3\%$$

$$CO_2 = \frac{0.275}{2.17} \times 100 = 12\ 7\%$$

Example 2. Analysis of the mine air from the worked-out area of a coal mine, isolated by stoppings, is.

0.72% O_2, 7 47% CH_4; 11 03% CO_2; 80 78 % N_2

The differential composition of this air is

Atmospheric air	. . . .	3 44%
Methane	. . . .	7.47%
Dead air	. . . .	89 09%
Total	. . . .	*100.00%*

1-2. THE MAIN COMPONENTS OF MINE AIR

1-2.1 Oxygen, O_2

Oxygen is a colourless, odourless and tasteless gas, slightly soluble in water. Its specific gravity is about 1.11 relative to air. Oxygen is a highly active element: it readily combines with a large number of gaseous elements and compounds. It is essential for all forms of life, and for combustion.

The *main* causes of a reduction in the amount of oxygen in the mine air are: (a) the slow oxidation of various organic and mineral substances (timber supports, rocks, minerals), mine fires and explosions of methane-air mixture or dust; (b) the addition to mine air,

The amount of oxygen taken up by the organism through the lungs from the air depends on the partial pressure of the oxygen in the air breathed in. Oxygen is best used by the blood at a partial pressure of about 160 mm of mercury Under an atmospheric pressure of 760 mm Hg this corresponds to the normal oxygen content in the air (about 21 per cent). But owing to the adaptability of the human body we can even breathe air under a partial pressure of 65 to 90 mm Hg. Under an atmospheric pressure of 760 mm of mercury this corresponds to an oxygen content of 9-12 per cent by volume; at this concentration. breathing is possible provided only that the remaining amount (88 to 91 per cent) consists of nitrogen or a similar inert gas and that the transition from the normal to the reduced oxygen content is gradual. It should also be remembered that even at an oxygen content of 12 per cent breathing is difficult, and when the concentration of oxygen falls off to 9 per cent, unconsciousness can rapidly occur, with death due to oxygen starvation (anoxaemia).

Man can breathe pure oxygen or air rich in oxygen without difficulty or harm provided that the pressure does not exceed 2 atmospheres, but this air will irritate the lungs if it is inhaled for a long time, say, some tens of hours together.

The safety regulations in the USSR require the amount of oxygen in the air of work places underground to be maintained at or above 20 per cent.

In other countries, the minimum allowable percentage of oxygen in mines is 19 per cent

In underground conditions, beginning at an oxygen content of 17 per cent, particularly during work, shortness of breath and palpitation are likely to be felt.

In underground workings ventilated badly or not ventilated at all, or after blasting or underground fires, the amount of oxygen may drop to 1-3 per cent. Breathing of such air is liable to cause immediate unconsciousness and death. For this reason an inspection of worked-out areas and unventilated dead ends should involve the testing of the air with a flame lamp or a special instrument.

If the lowering of the oxygen content is not accompanied by an increase in the amount of carbon dioxide or other gases, men become accustomed to it and can breathe air with a rather low oxygen content. For example, it is known that mountain dwellers, living at a height of 4,000-5,000 metres above sea level, where the partial pressure of oxygen is 85-96 mm Hg, have excellent health. This is explained by the fact that the human body can with time adapt itself to various environmental conditions.

This adaptability of man is so high that at smaller altitudes, e g. about 1,500 metres above sea level, even people from the plains do not notice that the atmospheric pressure is about 650 mm instead

of 760 mm of mercury, consequently the oxygen partial pressure is equal to $159 \times \frac{650}{760} = 136$ mm Hg, and the amount of oxygen in air (by weight) is equivalent to about 18 per cent at sea level $\left(20.96 \times \frac{650}{760} \approx 18\%\right)$.

In deep mines, on the other hand, the air is denser than at the earth's surface, and the partial pressure of oxygen may correspond to normal even at a reduced oxygen content in the air. This occurs, for example, in mines at a depth of 800 metres with a barometric pressure of 816 mm, where the oxygen partial pressure is 152 mm, even at a percentage oxygen content of only 18.7 per cent. From this we should not conclude that the minimum statutory oxygen content in the air in Soviet mines (20 per cent) is a high one: it is necessary because in underground conditions there is no sunlight which is of a high value for health

In mines 1,500-2,000 metres deep the air pressure is about 15-30 per cent higher than on the earth's surface, but this does not noticeably affect the gas exchange in the human body. Miners and divers have sometimes to breathe air under pressures considerably higher than normal (up to 4-5 atmospheres). This takes place, for example, in the sinking of mine shafts in heavily water-bearing rock, when men work in caissons and the air is forced into the caisson in the shaft at such a pressure that the water is pushed out into the ground and the face remains dry.

When men work at pressures higher than atmospheric, the blood and the tissues of the body begin to absorb nitrogen. If the pressure reverts abruptly to normal, the nitrogen will be removed from the body, and this may cause a number of painful and extremely dangerous conditions in the body (earache, breakage of the eardrum, headache, itching, giddiness rheumatism in the joints, paralysis, and even death).

Because of the dangers arising while working in caissons, special rules have been worked out, which must be strictly observed These rules lay down the precise sequence and rate of raising and lowering of the air pressure when men go in or out of the caisson.

It is quite obvious that to maintain the normal gaseous interchange in the human body, not only the necessary quality but also the necessary quantity of air must be provided.

In a state of rest and in normal atmospheric conditions of temperature and pressure man breathes 10-15 times per minute, thus breathing in and out some 5-7 litres of air.

In movement or under nervous tension, or during work the amount of air inhaled increases to 20 litres, and in particularly heavy work it may reach 40 litres per minute. In reality, however, to maintain

normal gas exchange, the amount of air required should considerably exceed that indicated owing to the peculiarities of the process mentioned above and because the air breathed in inevitably contains some of the air expired. Experience has shown that in mines where human breathing is the only source of air pollution, the fresh air supply per man should be not less than 1 m^3/min.

To test the mine air for oxygen content, use may be made of the miner's flame lamp. When the oxygen content drops to that below normal, the flame diminishes and may be extinguished. For example,

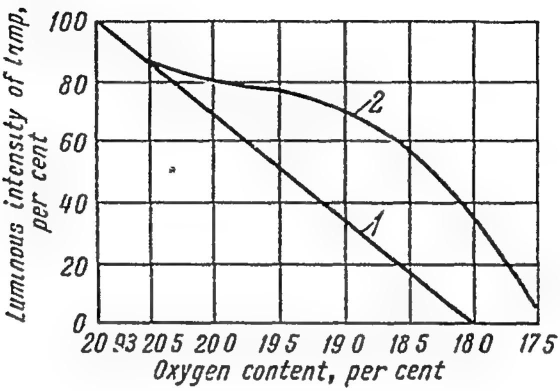

Fig 1-1 Luminous intensity of a flame safety lamp versus oxygen content of the air
1—wick at constant height, *2*—wick progressively raised

the miner's flame safety lamp is extinguished at an oxygen content of 16 5-17.5 per cent, and at 19 per cent its luminous intensity drops to 2/3 that in normal air.

Approximately every 0.1 per cent drop in the oxygen content reduces the light from the lamp by 3 5 per cent (Fig. 1-1).

A more accurate estimate of the amount of oxygen in mine air than that obtained from the flame lamp is provided by a portable gas analyser, e.g. the gas analyser designed by MakNII (the State Research Institute for Mine Safety in the Donets Basin coalfield).

The MakNII instrument (Fig. 1-2) works on the principle of absorption of the air oxygen by copper in an ammoniacal solution of ammonium chloride and of measuring the corresponding reduction of the pressure in the reaction chamber. The instrument consists of the following parts an air pump (a rubber bulb) *1*; a container *2* with a chemical absorbent for CO_2; a reaction chamber *3* consisting of a vessel *4* filled with a solution of ammonium chloride and ammonia, in which a copper spiral *5* is immersed, a reaction chamber *6*, which during the air test is closed by a rubber bung mounted on

a rod; and a diaphragm pressure gauge 7, which is calibrated to indicate the percentage oxygen content in the air being tested

To take an air sample, valve 8 is opened and the air is brought by 8-10 squeezes of the bulb via the CO_2-absorption container into reaction chamber 6 By means of a small handle not shown in the drawing, the copper spiral is drawn out of the solution into the reaction chamber and held there for about 5 minutes, following which the spiral is lowered into vessel 4, and the chamber is connected

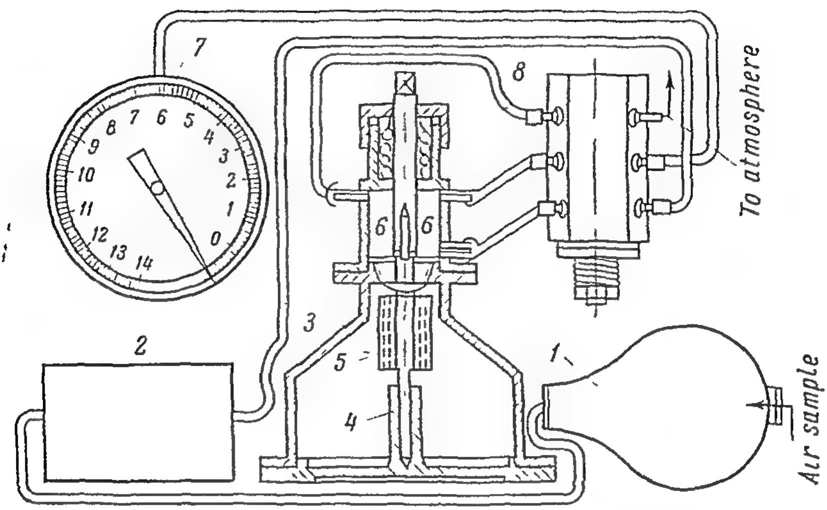

Fig 1-2 MakNII instrument for determining the oxygen content of mine air

to the manometer The manometer needle indicates the oxygen content in the air sample to an accuracy of 0.1 per cent The CO_2 absorbent lasts for 100 estimations, after which it must be changed.

1-2 2 *Nitrogen*, N_2

Nitrogen is a colourless, odourless and tasteless gas, slightly soluble in water. Its specific gravity is 0.97 relative to air, it supports neither breathing nor combustion

The sources of nitrogen in mines include

(1) the decomposition of organic substances,

(2) explosions (for example, 1 kg of nitroglycerine releases 640 litres of gas including 135 litres of nitrogen),

(3) emissions, either mixed with methane or as pure nitrogen, from cracks in rock or coal, including tertiary coal,

(4) emissions of dead air.

In productive mine workings the nitrogen content varies but slightly, in sealed old workings it may vary by 10, 20 or more per cent.

1-2 3 *Carbon Dioxide*, CO_2

Carbon dioxide (carbonic acid) is a colourless gas with a slightly acidic taste and odour. Its specific gravity is 1.52 relative to air. It is slightly poisonous. Carbon dioxide causes irritation of the mucous membranes of the eyes, mouth and nose, and at 5-10 per cent it produces a slight sensation of burning. At very high concentrations (over 20 per cent) CO_2 produces a feeling of burning on exposed human skin It supports neither combustion nor breathing, and is readily absorbed by water at 0°C and an air pressure of 760 mm Hg, 180 volumes of CO_2 are dissolved in 100 volumes of water

The main sources of CO_2 gas in mines are

(a) the decomposition of organic substances, mainly the rotting of timbering;

(b) the disintegration of rocks, whether of organic or mineral origin; the slow oxidation of coal and the disintegration of carbonate rocks by acid mine waters According to some authors the disintegration of dolomites by sulphuric acid resulting from the decomposition of pyrite can yield 1,000 m³ of carbon dioxide per day;

(c) the emission of carbon dioxide as such from rocks or coal, with or without methane.

Methane emitted from blowers (see below) very frequently contains carbon dioxide In some coal mines there is a constant and fairly uniform emission of pure CO_2 from the floor of the coal seam, as a consequence of which men have been known to suffocate while undercutting the coal.

In some coalfields CO_2 emissions occur extremely violently, in the form of sudden outbursts with large quantities of small coal. For example, in the Gard coalfield in the south of France (at the mines of Rochebelle and Fontane), sudden outbursts of large quantities of carbon dioxide take place in the face from time to time

In Soviet coal mines, sudden outbursts of CO_2 have not so far been observed, but in the mines of the Moscow region coalfield sudden changes in the barometric pressure (usually a drop in pressure), result in the appearance of large quantities of the gas in the mine which makes work difficult. This very complicated physicochemical process is explained as follows.

Tests have shown that if some Moscow coal is placed in a tightly closed vessel full of air, the oxygen is rapidly absorbed with the formation of CO_2; after some time the vessel contains no more than

a fraction of 1% of oxygen. If the vessel is refilled with fresh air, the oxygen is again absorbed

Apparently something similar happens in the working places and wastes of the Moscow region coalfields where in the innumerable cracks in the coal seams or in the coal left in the wastes, the oxygen is absorbed and there is a gradually increasing accumulation of carbon dioxide.

With changing barometric pressure underground there is inevitably some shifting of the mass of air in the workings and waste areas, and the accumulated CO_2 is released, gradually migrating from the wastes into the working places

The main practical measures against these gas emissions in the Moscow coalfield are

(a) careful sealing of the wastes;

(b) installation of a second (stand-by) fan enabling both fans to be used together when the gas is emitted

When a mine being sunk passes through mineral springs the liberation of CO_2 is sometimes very great, up to several thousand m^3 per day.

Very large quantities of CO_2 are formed in explosions either of methane or of dust and in mine fires.

Secondary sources of carbon dioxide in the mine air are

(a) The breathing of men and animals. The air breathed out by man ordinarily contains about 4 per cent of CO_2 The absolute quantity varies in very wide limits and depends generally on those factors which govern the variations in the human oxygen requirement. In underground workings the miner gives out, on the average, 50 litres of CO_2 per hour. A horse breathes out about 6 to 8 times as much

(b) The burning of lamps A flame lamp burns about 6-7 grams of fuel per hour, forming about 10 litres of CO_2 during this time.

(c) Internal-combustion engines used in locomotives and sometimes also for driving pumps or other machines We can say approximately that petrol engines produce about 60 litres of CO_2 per horsepower-hour.*

(d) Explosives used in blasting operations Explosion of 1 kilogram of gelatinous dynamite releases about 250 litres of CO_2

The breathing of men and animals, the burning of lamps, and blasting are secondary sources of carbon dioxide underground, at least in coal mines. Usually these sources yield a total of only 1/10 to 1/20 of all the carbon dioxide formed underground, but there are instances where this ratio rises to 1/3.

---

* For more detailed data on this problem see the section on carbon monoxide

Observations in 17 Donets Basin coal mines carried out by D. E. Borisov and A. A. Skochinsky established the variations in this ratio as between 1/10 and 1/3.

We can state that for metal mines it is about 1/3.

As far as the absolute quantity (Q, m^3) of CO_2 formed per day underground is concerned, and the quantity produced per ton of daily output (q, m^3), these depend mainly on: (a) the properties and the tendency of the mineral and rock being excavated to release CO_2; (b) their liability to oxidize with the formation of CO_2; (c) the dimensions of the mine; and (d) the age of the mine, which is the most important factor. The older the mine the larger are Q and q because the larger are the volumes of worked-out areas in which large quantities of carbon dioxide are released from various sources, in particular, from the rotting of timber, etc.

It has been established for coal mines (G. D. Lidin) that the amount of CO_2 in the mine increases considerably (although *not proportionately*) with increase in the quantity of air supplied per unit time.

Examination of 19 Donets Basin coal mines and 5 Moscow coal mines gave the following results:

Donets Basin coalfield

Mines working coking coal .	from 1.5 to 7.0 m^3/ton of daily output
Mines working gas coal with a long flame	from 3.0 to 4.5 m^3/ton of daily output
Anthracite mines .	from 3.5 to 14.5 m^3/ton of daily output

Moscow region coalfield

Brown coals	from 3.0 to 7.0 m^3/ton of daily output

The largest emissions of CO_2 have been observed in the Artem mine at the town of Shakhty: from 13.5 to 14.5 m^3/ton of daily output with a mean output of 2,500 tons per day. However, it should be pointed out that in the old Donets Basin mines with vast waste areas, q may be considerably larger than this.

A later, systematic examination by MakNII of 285 Donets Basin coal mines for their carbon dioxide emissions showed considerably larger values in some mines, for example. 51 mines with emissions of more than 15 m^3/ton of daily output, including four mines with 39, 45, 57, and 89 m^3.

According to the data obtained by V. B. Komarov in a number of mines in the Karaganda coalfield, q for CO_2 was from 6 to 11 m^3/ton of daily output.

The average content of CO_2 in the mine air depends, other conditions being equal, on the effectiveness of mine ventilation In well ventilated mines it usually does not exceed 0.25-0.30 per cent*, but in those which are poorly ventilated it varies from 0.75 to one per cent

The inability of carbon dioxide to support combustion enables a large concentration to be determined in the mine by the extinction or bad burning of lamps It is impossible to say exactly at what content of CO_2 miner's lamps begin to burn badly or are extinguished because these phenomena depend not only on the CO_2 content but also on the oxygen content and a number of other factors. In still air, the lamp is extinguished at a lower CO_2 content than in moving air When a lamp burns in a small space, a larger CO_2 content is needed to extinguish it than when the space is larger.

The hotter the flame, the more it resists the action of CO_2.

In still air the flame of a safety lamp begins to die down at a content of 3-4 per cent CO_2, if the air is moving the lamp will burn at a higher CO_2 content

Carbon dioxide plays an important role in the gas interchange taking place in the human body, *by stimulating breathing*. It has been shown that the rate and volume of respiration depend on the CO_2 content in the alveoli

An increase in the alveolar content of CO_2 of only 0.2 per cent doubles the ventilation rate of the lungs

It is well known that an acceleration of the walking pace or the intensity of working will increase the frequency and depth of breathing This results from the fact that the gas exchange in the body increases with an increase in the amount of CO_2 formed in it; the blood, becoming more acid, irritates the breathing centre, intensifying the respiration When the movement slows down or the intensity of work diminishes, the effect is reversed

At higher concentrations of CO_2 in the air breathed in, this gas becomes poisonous. It then becomes difficult for the carbon dioxide to be released from the venous (stale) blood and for oxygen to be absorbed by the arterial blood At very high CO_2 contents in the air breathed in, the CO_2 partial pressure becomes so large that the arterial blood begins to absorb CO_2 with the O_2 The blood therefore becomes enriched in carbon dioxide and impoverished in oxygen (oxygen starvation) and the CO_2 poisons the nerve centres The resulting illness can become fatal

It is extremely difficult to establish accurate general standards for CO_2 contents in mine air, which are harmless or unhealthy or

* The carbon dioxide content of the air in individual workings particularly dead ends or old workings, may be considerably higher even in well ventilated mines

fatal, because the reaction of the body to excess CO_2 in the air is even more complicated than that of the flame and in addition the human body is extremely adaptable.

According to current Soviet standards for dwellings, workshops and so on, the CO_2 content becomes harmful when it exceeds 0 1 per cent. It is impossible to apply this standard unreservedly to mines because in mine air, as we have seen, only part (often a negligible part) of the CO_2 is of physiological origin (formed by the breathing of men and animals). The carbon dioxide resulting from breathing is more harmful than that of mineral origin because the former, according to certain physiologists, contains other poisonous vapours and gases eliminated from the body during breathing and from the skin.

According to Soviet Safety Regulations, the content of CO_2 in productive mine workings should not exceed 0 5 per cent by volume. whether in coal mines or in metal mines.

At carbon dioxide concentrations higher than 1 per cent, man begins to breathe in noticeably more air than in the case of normal air. Thus, with 3 per cent CO_2 in the air the breathing is doubled even in a state of rest At rest, however, this is not harmful but if the man is working, his breathing becomes even more faster, with resulting fatigue. At 5 per cent CO_2 content the breathing is tripled and becomes extremely heavy; at 6% there is dyspnoea and weakness; at a concentration of 10% or more, fainting can occur and 20-25% may be fatal. These symptoms are even more pronounced if, together with the increased CO_2 content, there is a reduced oxygen content. If the oxygen content remains high (above 20 per cent), the effect of the carbon dioxide is much less severe.

Since it is considerably heavier than air, carbon dioxide can collect at the floor and flow to the lowest part of inclined workings, easily forming a dangerous concentration if the ventilation is poor. This must always be remembered when approaching a badly ventilated place, particularly an old working, especially after an interruption in work. In these cases a preliminary test of the air should be carried out, for example, with flame safety lamp lowered on the end of a rod or a rope. If the lamp is extinguished, then entry into the gassy place is only possible with a respirator.

The carbon dioxide content in mine air is detected or determined by:

(1) chemical analysis of the air, in the laboratory,
(2) testing with flame safety lamps;
(3) testing by means of various reagents;
(4) special portable gas analysers

One of the methods of testing by reagents is as follows. In a flask of 110 cm^3 capacity, 10 cm^3 of a violet-red solution of phenolphthal-

ein ($C_{20}H_{14}O_4$) and soda is poured. The flask is then closed with a bung, through which two glass tubes are passed, one short and the other reaching nearly down to the bottom of the flask. The top end of the long tube is inserted into a rubber hose with a rubber bulb of 70 cm³ capacity fitted with a valve. By squeezing the bulb, the air sample is forced through the solution. In the presence of CO_2

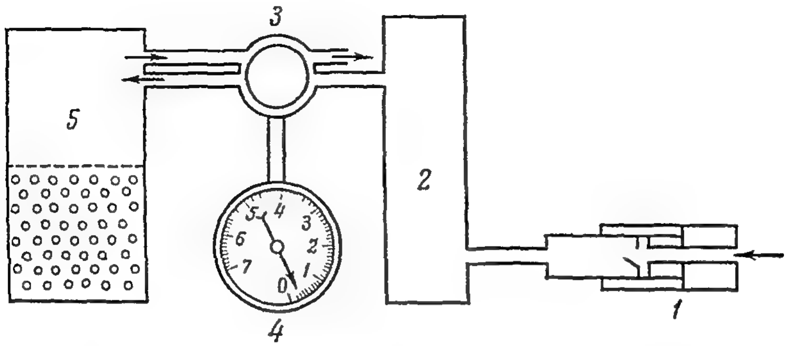

Fig 1-3 MakNII instrument for determining the CO_2 content of mine air

the solution is bleached. The number of times the rubber bulb is squeezed gives an indication of the CO_2 content in the air:

Number of squeezes of bulb	2-3	4-6	6-7	8-10
Approximate CO_2 content for bleaching the solution, per cent	0.30-0.25	0.20	0.15	0 10

The phenolphthalein solution should be prepared in the following way dissolve 5 3 g of anhydrous soda (or 14.3 cm³ of normal solution of crystalline soda) in 1 litre of freshly boiled and cooled water, to this solution add 0 1 g of phenolphthalein, a dark violet liquid results, to make an air test take 2 cm³ of this liquid and add it to 100 cm³ of distilled, freshly boiled and cooled water

When the CO_2 content exceeds 0 25-0 30%, either fresh air must be added to the sample or the reagent concentration must be increased

The MakNII portable CO_2 analyser (Fig 1-3) works by absorbing the carbon dioxide from the air sample by a solid chemical absorbent (XIIII) and measuring the resulting pressure reduction in the chamber.

The apparatus (Fig 1-3), placed in a small aluminium box, consists of the following parts a pump (*1*), a humidifier (*2*) filled with wet pumice; a three-way cock (*3*), a diaphragm manometer (*4*); and a reaction chamber (*5*) full of the dry chemical absorbent The

sequence of work with the gas analyser is as follows· with some 6-8 strokes of the pump the air sample is brought into the reaction chamber, thereby passing through the humidifier; the reaction chamber is shut off by turning the valve and the air sample is held in the sealed chamber for 3 minutes, after which the valve is again turned to connect the chamber to the manometer. The manometer scale shows the CO_2 content (in per cent) of the air sample According to MakNII the accuracy is 0.1% CO_2, one charge of chemical absorbent is enough for 500 determinations. A portable type ШИ-5 interferometer for determining the carbon dioxide content directly underground has recently come into use.

1-3. THE MAIN HARMFUL IMPURITIES IN MINE AIR (POISONOUS OR EXPLOSIVE)

1-3.1 Carbon Monoxide, CO

Carbon monoxide is a colourless, tasteless and odourless gas, having a specific gravity of 0.97 relative to air. It is slightly soluble in water 3 volumes of the gas dissolve in 100 volumes of water in ordinary mine conditions. Carbon monoxide is combustible (its flame is dark blue to pale blue) and explosive. When it is formed or burns the following quantities of heat are liberated:

$$C+O=CO+2{,}420 \text{ kcal per kg of carbon}$$

$$CO+O=CO_2+5{,}660 \text{ kcal per kg of carbon}$$

At ordinary temperature and pressure, carbon monoxide is fairly inert chemically, though explosive in mixtures with air from 13-16 up to 75% (in normal conditions, that is at a pressure of about 1 atmosphere and room temperature). The strongest explosion of carbon monoxide takes place at a concentration of about 30% .The temperature of ignition of carbon monoxide and air mixtures is 630-810°C.

Carbon monoxide is poisonous because the haemoglobin of the blood, contained in the red corpuscles (erythrocytes) of the human body, combines much more easily with carbon monoxide than with oxygen (250 to 300 times more easily). Therefore if the air inhaled contains carbon monoxide, the blood absorbs carbon monoxide in preference to oxygen. Instead of oxyhaemoglobin (haemoglobin, Hb + oxygen, O_2 = HbO_2, oxyhaemoglobin), carboxyhaemoglobin, HbCO, begins to circulate in the blood, and oxygen starvation (anoxaemia) begins, which can be fatal if the CO content of the blood increases.

The reaction between oxygen, carbon monoxide and haemoglobin is reversible and can be expressed as follows.

$$HbO_2+CO \rightleftarrows HbCO+O_2$$

that is, carbon monoxide forces the oxygen out of oxyhaemoglobin, but in return, excess oxygen can force the carbon monoxide out of carboxyhaemoglobin

The largest quantity of carbon monoxide which can be absorbed by the blood of a man of medium build is about 300 cm^3

The degree of possible saturation of human blood with CO at various percentage contents of the gas in the air inhaled can be calculated in the following way

The air in the alveoli contains (apart from moisture) about 14% O_2 From this it follows that, since the haemoglobin combines with CO some 250 times more rapidly than with O_2 and the oxygen pressure in the blood is about the same as in the alveoli, the blood acquires equal proportions of carbon monoxide and oxygen when the carbon monoxide content in the air inhaled is equal to $\frac{14}{250} =$ $= 0\ 056\%$, Table 1-1 shows other degrees of saturation that will result from the concentrations of carbon monoxide given

TABLE 1-1 **Degree of Saturation of the Blood with Carbon Monoxide**

Proportion in the blood		Degree of saturation of the blood with CO	CO concentration at which this condition of the blood is possible, per cent
HbCO	HbO_2		
1	3	1/4	0 014
1	2	1/3	0 028
1	1	1/2	0 056
2	1	2/3	0 112
3	1	3/4	0 168
4	1	4/5	0 224

Note Hb indicates haemoglobin

Three degrees of acute (i e not chronic) poisoning by carbon monoxide are recognized (Table 1-2)

1 Slight poisoning—buzzing in ears, headache, giddiness, and palpitation

2 Severe poisoning, with all the symptoms of slight poisoning and in addition loss of the power to move and dulling of the consciousness

3 Fatal poisoning—unconsciousness, convulsive movements, death

TABLE 1-2 **Symptoms of Acute Carbon Monoxide Poisoning**

Percentage haemoglobin in the blood, combined with carbon monoxide, per cent	Symptoms of acute (non-chronic) poisoning
10	Not noticeable apart from slight shortness of breath upon intensive muscular exertion
20	Not noticeable apart from slight shortness of breath and palpitation even upon slight muscular exertion
30	Headache, irritability, quick fatigue, dulling of the intellect, vomiting
40-50	Intense headache, weakness, fainting upon muscular exertion
60-70	Unconsciousness, with death after a time (usually short)
80 and over	Immediate death

The degree and rapidity of CO poisoning depend on

(a) the concentration of carbon monoxide in the air,

(b) the quantity of air inhaled per unit time;

(c) the rapidity of blood circulation,

(d) whether the poisoned air is breathed in continuously, or alternately with fresh air, according to Novitsky, cigar smoke contains up to 5-7% of carbon monoxide but since the smoker alternates his draws of tobacco smoke with breathing in of fresh air, the poisoning is not severe

It is therefore difficult to give numerical values for the relation between the carbon monoxide content of the air, the time during which this air is inhaled, and the resulting degree of poisoning.

Table 1-3 shows four different categories of toxic concentrations of carbon monoxide* for a man in a state of rest

---

* To convert gas concentrations expressed in volume percentages to weight contents (mg per litre) and vice versa, the following equation can be used

$$C\% = \frac{2.4}{10M} \times C, \text{ mg per litre}$$

where M = mole (the gram-molecule, the weight of which in grams is numerically equal to the molecular weight of a gas) M for carbon monoxide is 28, for nitric oxide 30 01, for nitrogen peroxide 46 01, for sulphur dioxide 64 06 for hydrogen sulphide 34 08.

TABLE 1-3 Categories of Toxic Carbon Monoxide Concentrations

Degree of poisoning	Durations	Carbon monoxide concentration	
		mg/litre	percentage by volume at 10°C and 760 mm Hg
1 Only minor symptoms	Some hours	0.2	0.0016
2. Minor poisoning	One hour or less	0 6	0 048
3 Severe poisoning	30-60 minutes	1 6	0.128
4 Fatal poisoning	Very short time	5 0	0 4

When the CO content in the air is 1%, a man will lose consciousness after several breaths

Prolonged (several hours per day) work in an atmosphere containing only 0 01% carbon monoxide will result in chronic poisoning with serious consequences. A harmless concentration of the gas is considered to be 0.0016% maximum For short periods (30 minutes) of breathing of carbon monoxide the safe limit can be raised to 0 02% by volume.

Carbon monoxide is formed in those instances when CO_2 is reduced by contact with reducing agents such as carbon, metals, steam, etc When carbon is burned, carbon dioxide is usually first formed, but if the carbon dioxide is passed through or over a layer of hot carbon, it decomposes according to the equation. $CO_2 + C = 2CO$ The hot gauze of a miner's lamp exerts the same effect on carbon dioxide The burning of carbonaceous material can also go directly through to carbon monoxide; it has also been established that the gas is formed in small quantities by the slow oxidation of coal at temperatures of 20-30°C.

Among other gases (mainly CH_4, CO_2 and N_2), some coals also contain carbon monoxide (up to 14-15 cm³ per kg of coal) and this is liberated together with the other gases, but usually in very small quantities

D. I Kovarsky has shown that carbon monoxide is permanently present, though in insignificant quantities, in the workings of the Moscow coalfield

However, the main sources of CO in mines are. (a) fires; (b) explosions of methane or coal dust, in particular the latter, (c) blasting

operations. Carbon monoxide is also formed underground when internal-combustion engines are used.

Any estimation of the quantity of CO formed during underground work with explosives is exceedingly complicated and has not yet been fully cleared up, because it has been shown that the amount of carbon monoxide liberated depends not only on the ingredients of the explosive and the manner of confinement and detonation, but also on the completeness of detonation and even on the medium in which the shot is fired. For the practical purposes of calculations, it is accepted that 1 kg of explosive used underground releases 40 litres of so-called conventional carbon monoxide (CO plus 6.5 times the NO_2 content).

How easily very dangerous quantities of carbon monoxide can be formed underground even in the case of a very small fire can be seen from the following calculations which are concerned with the burning of timber: 1 m³ of dry wood weighs on the average 750 kg; the average composition of the wood is 40% C (including 1.5% ash), 40% of chemically bound water and 20% of uncombined water. Twelve parts by weight of carbon combining with 16 parts of O_2 yield 28 parts of CO.

If the wood burns in conditions such that all the carbon is converted to CO, then 1 m³ yields:

$$\frac{750\,(40-1.5)\,28}{100\times 12}=673 \text{ kg CO}$$

or approximately 570 m³ CO, because at 15°C and 1 atmosphere 1 m³ of CO weighs about 1.18 kg. The volume of a pair of props 2.1 m long and a bar 2.4 m long (both are of 18 cm in diameter), forming one timber frame, is about 0.17 m³.

Thus, a single timber frame includes enough carbon to form about 97 m³ of CO, and this quantity of carbon monoxide is enough to create a fatally poisonous atmosphere in a working 4-5 m² in cross section and more than 2 km long.

Underground experience has confirmed this conclusion.

Thus in the small metal mine of Snaefell in England the timber supports in the roadway were ignited by a lamp and 18 men were killed by carbon monoxide poisoning as a result. The fire spread along only 10 metres of the roadway. On this length, some of the timbers were completely burnt and some were partly burnt.

As we shall see below, ignitions of methane alone do not generally result in the formation of carbon monoxide, but if the ignition takes place in the presence of coal dust, the latter inevitably takes a part in the explosion and enriches the explosion products with carbon monoxide. Even more carbon monoxide is formed in explosions of dust alone. By a similar calculation to that above, it is easy to see

that 1 kg of coal dust containing 75% C, when burning in favourable conditions, can yield about 1 5 m³ of carbon monoxide

Blasting is also a source of carbon monoxide for the following reasons In the first place many explosives, even if they detonate completely, yield some considerable quantity of carbon monoxide

Secondly, in practice not every shot detonates completely, and when explosives detonate incompletely, carbon monoxide is formed even from those explosives which normally produce none. For example, nitroglycerine, when it detonates completely, yields CO_2, H_2O, N_2, and O_2, but when nitroglycerine burns, the resulting gases may consist of one third of carbon monoxide together with considerable quantities of oxides of nitrogen which are even more poisonous than CO *

Thirdly, CO is formed when shots are fired in coal or when the stemming material is small coal Instances are known of fatal CO poisoning when shot-holes in coal were charged with dynamite.

Special investigations were carried out for determining the quantities of poisonous gases released by internal-combustion engines (Table 1-4), which showed that these engines may be important

TABLE 1-4 **Quantities of Poisonous Gases Formed by Internal-combustion Engines**

Cylinder dimensions, mm	Number of cylinders	Speed, rpm	Maximum quantities of harmful gases formed at 16°C and 760 mm Hg, litres/min				Quantity of pure air needed to dilute the CO to 0 05% by volume (columns 4 and 6), m³/min	
			with good carburation		with poor carburation		with good carburation	with poor carburation
			CO	CO_2	CO	CO_2		
1	2	3	4	5	6	7	8	9
127 × 127	4	600	59	152	225	86	118	450
127 × 127	4	800	79	205	300	105	158	600
152 4 × 152 4	4	700	119	315	450	166	238	900
177 8 × 177 8	6	500	202	525	770	282	404	1,540
203 2 × 177 8	6	500	265	675	1,000	367	530	2,000

* Investigations carried out in the USSR have shown that the composition of the gases produced by shotfiring depend, among other factors, also on the medium in which the shot is fired Thus, for example, in experimental underground shotfiring it was shown that the same explosive, Ammonit No 2, in holes drilled in iron ore produced more than 5 litres of CO per 1 kg of explosive; in copper ore it gave 15 litres, in clay shales more than 20 litres and in coal more than 40 litres

sources of poisonous gas in the air not only underground but also on the surface, because the exhaust gases, apart from CO and NO_2, contain even more poisonous gases such as formaldehyde and acrolein.

The exhaust gas of a car contains from 3.5 to 7% of CO by volume.

To detect carbon monoxide in mine workings and to determine its concentration in mine air, use may be made of:

(1) *chemical analysis*, in the laboratory, of an air sample taken underground;

(2) *colorimetric detectors* which show the change of colour of various reagents under the action of carbon monoxide;

(3) *thermal detectors* which record the heat emitted by the oxidation of CO in the presence of a catalyst;

(4) small animals such as mice or birds (canaries).

Of the various reagents used for testing the mine air for carbon monoxide we can mention palladium chloride ($PdCl_2$) and iodine pentoxide (I_2O_5) as well as a solution of fresh human or animal blood.

The action of carbon monoxide on paper moistened with a 1-2% aqueous solution of $PdCl_2$ produces metallic palladium, blackening the paper

$$PdCl_2 + H_2O + CO = Pd + 2HCl + CO_2$$

According to the observations of Novitsky, the paper blackens:

at a content of 0.1% CO after 10 minutes
at a content of 0.2% CO after 6 minutes
at a content of 0.5% CO after 4 minutes
at a content of 1.0% CO after 1 minute.

In practice palladium chloride ($PdCl_2$) is provided in small sealed ampoules, 40 mm long, 5 mm in diameter, which are easily crushed. They are filled with a solution of $PdCl_2$ (1-2%) in a mixture of water and acetone. When the ampoule is crushed, a cotton wool wad is moistened with the $PdCl_2$ and is held in the poisoned air for ten minutes. The CO concentration is established by comparing the colour of the moistened cotton wool with a table of standard tints. In using such ampoules, however, we must remember that the colour of the cotton wool moistened with $PdCl_2$ solution is also changed by hydrogen sulphide, hydrogen, ethylene, and gasoline vapour.

Figure 1-4 shows the iodine gas analyser for carbon monoxide.

The action of this gas analyser is based on the reaction of iodine pentoxide with carbon monoxide.

$$I_2O_5 + 5CO = I_2 + 5CO_2$$

The main parts of the instrument are: two tubes, one of which is a filter filled with activated charcoal *1*, to clean the air of H_2S, SO_2, and other gases which can act on iodine pentoxide in the same way as carbon monoxide; the second tube *2*, the detector, both reveals the presence and determines the content of CO in air; it is charged with pieces of pumice (hoolamite) soaked in iodine pentoxide, which have been subjected to the action of fuming sulphuric acid.* When the air is blown through this tube, the pieces of pumice, at first grey-white in colour, become green in the presence of CO, to a degree depending on its content in the air Another tube not

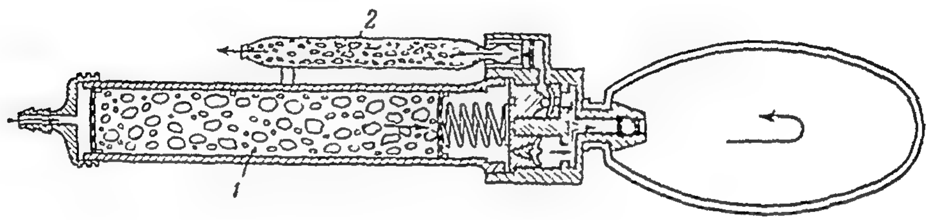

Fig 1-4 Iodine pentoxide analyser for carbon monoxide

shown in the drawing is mounted alongside the detector tube and acts as a scale The minimum concentration of carbon monoxide which is reliably indicated by this gas analyser is 0.05%.

A more sensitive and accurate method of determining carbon monoxide is the thermal method It is based on the increase of temperature of a special reagent called Hopcalite*, which catalytically oxidizes CO to CO_2 and is heated by the process

The sensitivity of stationary thermal gas analysers for carbon monoxide can reach 0.00001 per cent by volume, and of portable instruments about 0 001 per cent.

The *thermal carbon monoxide analyser* type OC-3 is shown diagrammatically in Fig 1-5. This portable instrument functions by measuring the heat released during the oxidation of CO to CO_2 in the presence of the catalyst Hopcalite

The instrument consists of a hand pump *1* (Fig. 1-5) with a dust-catching filter; an air receiver *2* of 0.4 litre capacity; a valve *3*, containers *4*, *5* and *6* filled with Carbogel, a reaction chamber *7* containing Hopcalite and a thermopile, an outlet container *8* also filled with Carbogel, a rotameter *9*, for determining the air speed; and a galvanometer *10* with a switch The air passes through the whole absorption system at a flow rate of 2 litres per min The con-

* The average composition of the mass is 12 29% I_2O_5, 51 89% H_2SO_4 and 35 82% pumice

* Hopcalite is a specially prepared mixture of copper oxide (CuO) and manganese peroxide (MnO_2)

tainers *4*, *5* and *6* are sealed metal cylinders filled with Carbogel to remove the moisture from the air. If moisture were to enter the reaction chamber it would reduce the reactivity of the Hopcalite.

The reaction chamber *7* is divided by an ebonite grid into two parts; in the upper part are the "cold" junctions of the thermopile and the conductors leading to the terminals; in the lower part are the "hot" junctions of the thermopile embedded in the Hopcalite. The container *8*, filled with carbogel, protects the Hopcalite from moisture in the air at the outlet from the chamber. The rate of air

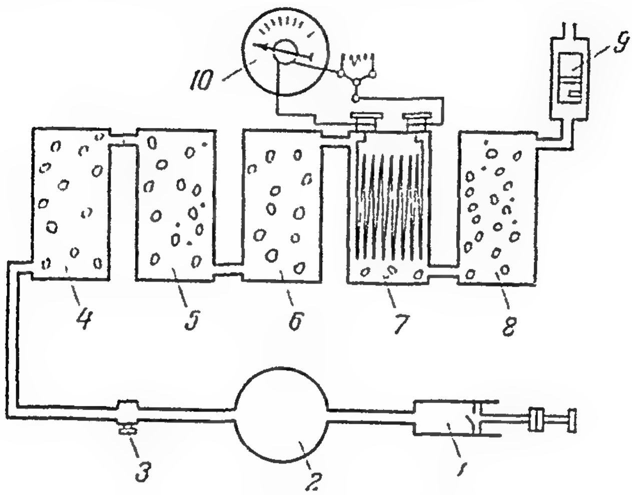

Fig. 1-5. Diagram of the OC-3 type analyser

flow through the whole system is regulated by the rotameter *9*. The galvanometer *10* measures the thermo-electromotive force of the thermopile; at high concentrations of carbon monoxide, a switch is used to bring resistances into the galvanometer circuit.

Attached to the lid of the body of the instrument is a table used to convert the galvanometer readings to the percentage CO contents.

For three to four minutes before each measurement, fresh air must be blown through the instrument; during this time the rotameter indicator must be held to the point which corresponds to an air flow of 2 litres per min.

During the measurement of the CO content, the air pump pushes the air sample through the absorption system of the instrument for 5 minutes, maintaining a flow rate not exceeding 2 litres per min. The galvanometer reading is then taken and the CO concentration is read off the table.

One charge of the containers lasts for 50-60 measurements.

The instrument determines carbon monoxide contents within the range of from 0.002 to 0.2% with an accuracy of 0.002%.

The chemical gas detector ГХ-1 operates on the principle of changing the colour of a solid reagent contained in the detector tube through which the air sample is passed. The instrument detects carbon monoxide at its concentration of 0.0005 to 0.36%.*

1-3.2 *Hydrogen Sulphide*, H_2S

Hydrogen sulphide is a colourless gas with a characteristic odour of rotten eggs, and a sweetish taste; its specific gravity is 1.19. It is a flammable gas and forms an explosive mixture with air at a concentration of 6% Hydrogen sulphide is readily soluble in water; at 1 atmosphere and 15°C, 1 litre of water dissolves 3.23 litres of H_2S. It is extremely poisonous and irritates the mucous membranes of the eyes and respiratory passages as well as the nervous system.

The toxicity of hydrogen sulphide is shown in Table 1-5, based on the observations.

TABLE 1-5 Toxicity of Hydrogen Sulphide

Concentration		Effect on human body after duration stated
per cent, by volume	mg/litre	
0 01	0.14	Slight poisoning after several hours
0 02	0 28	One hour without serious consequences
0.05	0 7	Serious poisoning after 30-60 minutes
0.10	1.4	Death quickly occurs

The main sources of hydrogen sulphide underground are:

(a) the rotting of organic substances, particularly wood; it is therefore dangerous to go into old workings where there is water, since there may be accumulations of the gas;

---

* The oldest method of determining the presence of CO, which is not yet obsolete, particularly in geological prospecting, is to use small warm-blooded animals

As has been pointed out, the rate of poisoning depends on the frequency of pulsation of blood In man the pulse rate is 60-70 strokes per minute, but in small warm-blooded animals and birds (mice and canaries) it reaches some hundreds per minute These animals are therefore poisoned by carbon monoxide very much more quickly than man, and it is possible to use them as carbon monoxide detectors White mice are more sensitive to carbon monoxide than grey mice Of the small birds (doves, sparrows, canaries) canaries are the most sensitive

(b) the decomposition of pyrite, gypsum, etc., by water:

$$FeS_2 + 2H_2O = Fe(OH)_2 + H_2S + S$$

$$CaS + H_2O + CO_2 = CaCO_3 + H_2S$$

(c) accumulations in cracks and hollows in the rock and mineral, particularly in rock salt as well as in petroleum or ozokerite mines;
(d) minerals intersected by workings;
(e) occasional release together with methane;
(f) blasting (incomplete detonation) and the burning of blasting fuse, as well as the burning of coal seams.

Particularly dangerous is the liberation of hydrogen sulphide from cracks and voids, because these may rapidly release a large quantity of the gas.

The presence of hydrogen sulphide in mine air is usually detected by its nauseating odour; it can be smelt even at a concentration of 0.0001 to 0 0002%, and at 0.02% it strongly irritates the mucous membranes of the eyes and throat in 5-10 minutes.

A more objective indicator of hydrogen sulphide is a paper strip moistened with a solution of lead acetate which darkens in the presence of the smallest trace of the gas. A darkening within 1 or 2 minutes indicates serious danger.

There are also special instruments for determining hydrogen sulphide content in the air.

The maximum allowable concentration of hydrogen sulphide in productive mine workings is 0.00066% by volume (0.01 mg per litre).

Since hydrogen sulphide is toxic and very soluble in water great care should be exercised in such places underground where the gas is smelt and there are water accumulations; if a prop or a piece of rock falls into the water, a fatally dangerous release of gas can occur.

If hydrogen sulphide is smelt underground, and particularly if it irritates the eyes or the throat, the gas must be neutralized by increasing the supply of clean air, saturated with finely sprayed water, into the area.

1-3 3 Sulphur Dioxide, SO_2

Sulphur dioxide is a colourless gas with a sharp taste and the commonly known pungent odour of burning sulphur. It strongly irritates the mucous membranes, particularly those of the eyes, even at concentrations in air as low as a few thousandths of one per cent, e.g. at 0.002%. It can be smelt even at a concentration

of 0 0005%. Sulphur dioxide is extremely poisonous a content of 0.05% is dangerous to life, even during a short time of exposure. Its specific gravity is 2 2. It is found underground in negligible quantities

Sulphur dioxide is formed in mine fires and in blasting operations if the explosives used contain sulphur or the rock with a considerable amount of sulphur is blasted. In addition, it is liberated in some mines together with methane; the attack on the mucous membranes of the eyes is extremely distressing to miners who call it the eye-eater. Sometimes hydrogen sulphide is liberated together with sulphur dioxide and hydrogen sulphide was therefore also formerly given the same name

The maximum allowable concentration of SO_2 in the atmosphere of productive mine workings is 0.00070% by volume (0.02 mg per litre).

The sulphur compounds H_2S and SO_2 are formed in dangerous quantities in sulphur mines when sulphur and copper pyrite ores rich in sulphur are being worked The pyrite is liable to ignite during blasting and in addition it forms a dust which is both flammable and explosive. Explosions in copper mines have been recorded

Tests on the dust from these ores have shown that it can be ignited from a charge of 75 g of 60% gelatinous dynamite; the dust explosion releases fairly large quantities of SO_2 and H_2S, up to 2-3 per cent; permissible explosives (for methane or coal dust) will ignite pyrite dust almost as easily as dynamite

Owing to its high specific gravity pyrite dust settles near the face and for this reason pyrite dust ignitions do not extend far from the face but are nevertheless a serious danger to miners.

The precautions to be taken in pyrite mines, apart from effective face ventilation after blasting, are as follows

(a) plentiful water spraying of the face and the nearby workings before blasting and before loading ore;

(b) careful stemming of the shot holes with wet clay;

(c) blasting to take place at the end of the shift according to a strict time-table, the men being removed from the face and from any workings in the return airway; there must be an easily audible signal to indicate the beginning of blasting and another one to indicate that the face is safe again;

(d) shotfirers must have gas masks available.

1-3 4 Gases Liberated by Explosives

Explosives used in blasting usually produce carbon monoxide (the poisonous properties of which are already known) and nitric oxide, NO Nitric oxide combining with the oxygen of the air is

converted to the more stable nitrogen peroxide, NO_2:

$$2NO + O_2 = 2NO_2$$

It is a heavy red-brown gas which gives its characteristic colour to the fumes of dynamite explosions, known to everyone who has been into the face shortly after blasting with dynamite. Nitrogen dioxide fumes are extremely poisonous, exerting a highly irritating effect on the mucous membranes of the eyes, nose, mouth and the bronchi and lungs.

Nitrogen oxides at concentrations over 0.025 per cent (0.5 mg/litre) may be fatal to a man in a short time. At lower concentrations of nitrogen oxides, a very serious danger to health may result if air containing these gases is carelessly inhaled deeply, since this may cause oedema of the lungs Such cases have repeatedly occurred, especially in metal mines.

A harmless concentration of NO_2 is considered to be 0.00025 per cent (0 005 mg/litre)

A characteristic feature of the action of nitrogen oxides on the human lungs is that the damage appears only after some time. A man, who is unsuspectingly doomed to death or the life of an invalid, returns home from work and in 20-30 hours may die because his lungs have been filled with fluid formed by the oedema. The nitrogen oxides are therefore the most dangerous of all the poisonous gases underground.

It is absolutely forbidden to go into a working place where the nitrous fumes have not been completely cleared after blasting.

The nitrogen oxides are readily absorbed by water vapour, and spraying water into the face greatly shortens the time needed to ventilate it after blasting.

One indicator for nitrogen oxides in mine air is a paper strip moistened with a solution of starch and potassium iodide; the paper rapidly blackens in the presence of nitrous fumes More accurate instruments for quantitative determination of nitrogen oxides (as well as of hydrogen sulphide and ammonia) in the mine atmosphere are those of types Гх-1 (chemical gas analyser) and УГ-1 developed by the All-Union Research Institute for Protection of Labour, Leningrad, under the Central Trade Union Organization

The instrument measures the length of a tinted column obtained when an air sample containing the gas being determined is passed through the detector tube. The length of the tinted column is proportional to the gas concentration in the air sample; the scale by which the length of the column is measured, is graduated in mg/litre.

The ranges of concentrations measured by the instrument are for hydrogen sulphide, 0 002-0 35 mg/litre (0 00013-0 023% by

volume); for nitrogen dioxide, 0 002-0 20 mg/litre (0.001-0.1% by volume).

The sensitivity of the instrument corresponds to the lowest limit of concentrations being determined; its error is 10-15% of the measured value.

The gas analyser, shown in Figs 1-6 and 1-7, consists of rubber bellows *1* with a feed spring *2*, an air intake used for all the gases

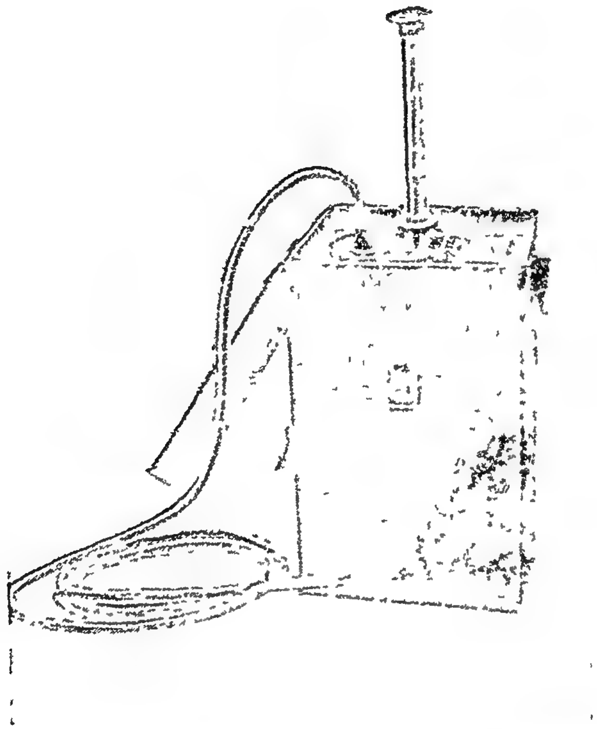

Fig 1-6 External view of the УГ-1 gas analyser

determined and a set of glass detector tubes The instrument weighs 2.2 kilograms.

The air is analysed for H_2S as follows the lid of the box is opened, the handle of the catch *5* in the guide bushing *6* is loosened; the rod *3* is placed in the bushing in such a way that its slot is in line with the catch *5* and is then pressed in until the upper slot *4*. in the rod *3*, is level with the catch The bellows are thus set to a fixed amount of compression

The detector tube *8* filled with a white powder is scraped free from its protective cap, compression by the pin eliminates the air gap between the column of powder and the tampon, and the tube

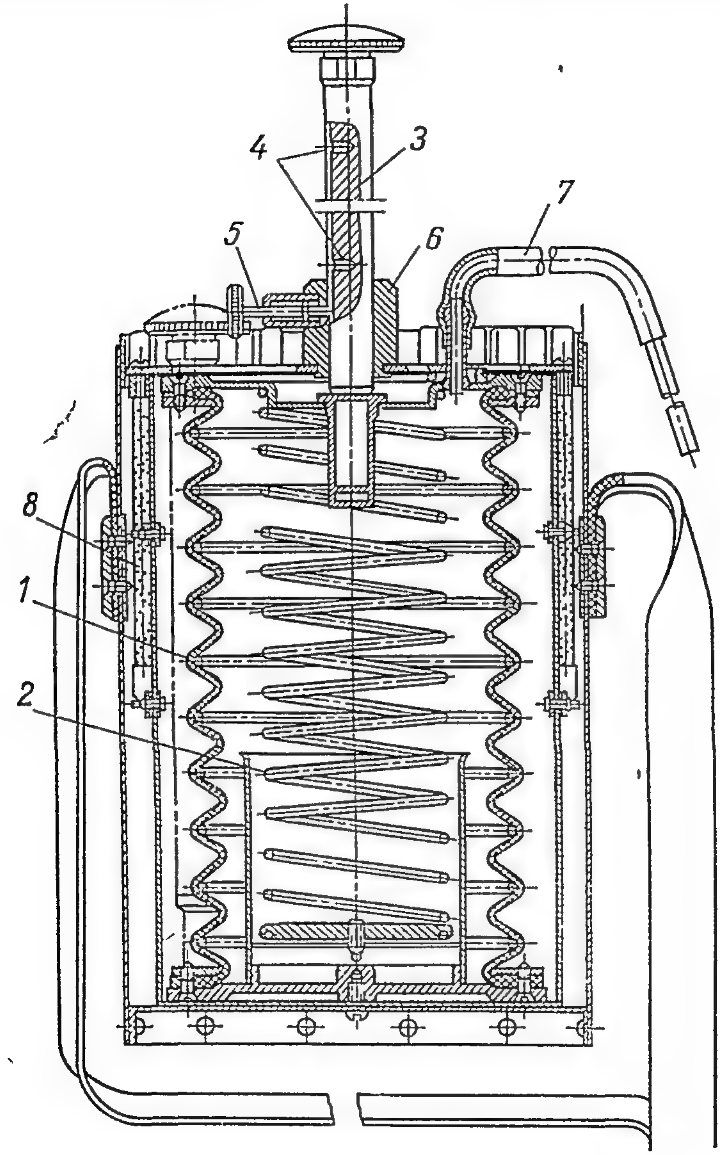

Fig. 1-7. Diagram of the УГ-1 gas analyser

is joined at one end to the rubber hose of the suction cap *7* of the bellows, the other end of the tube being introduced into the air to be tested The catch *5* is then released, the rod slowly rising and the air is drawn through the tube *8*. This is continued until the stop of the catch *5* automatically comes opposite the second, lower slot *4*

of the rod 3, when a click is heard and the rod stops moving. After 30 seconds to 1 minute the detector tube is released from the rubber connection and placed in the recess between the appropriate scales (marked H_2S 250 and H_2S 30) so that the beginning of the darkened part of the powder column is opposite the zero of the scale The figure opposite the upper end of the tinted part of the powder corresponds to the hydrogen sulphide content expressed in milligrams per litre of air analysed

Selection of the scale for determining the concentration depends on the volume of the air drawn through the detector tube. For example, if the amount of air is 250 cm³ (see above the rod slot) the correct concentration is read from the scale marked H_2S 250, if the air sample measures 30 cm³, the H_2S content is read off the scale marked H_2S 30

The glass detector tubes are 90-91 mm long, with an internal diameter of 2 5-2.6 mm, the powder indicators used for hydrogen sulphide and nitrogen dioxide are described in Table 1-6.

TABLE 1-6 The Properties of Indicator Powders

Gas indicated	Carrier	Grain size of carrier, mm	Reagent	White indicator powder changes colour, after action of gas, to
Hydrogen sulphide	Porcelain	0 25-0 30	$Pb(CH_3COO)_2$, $BaCl_2$, CH_3COOH	Dark brown
Nitrogen peroxide	Silica gel	0 20-0 30	Diphenylamine, NaCl, H_3PO_4, C_2H_5OH	Blue-green

1-3 5 The Gases in Compressed Air

The oil used for lubricating compressors vaporizes or decomposes (to the gases CO, CH_4, etc) because of the high temperature in the compressor and is carried along with the compressed air through the pipes to the point of air consumption in the mine These gases have repeatedly caused powerful explosions, destroying not only the compressor but also its building, and they have also caused poisoning of men

Every lubricant at a high enough temperature is decomposed into combustible gases; the decomposition of 20 g of lubricant can transform 1 m³ of air to an explosive mixture.

So that the air in the compressor does not become enriched with lubricant vapours and gaseous decomposition products (poisonous and explosive) it is recommended that:

(a) the cooling arrangements of the compressor be maintained in perfect condition,

(b) the compressor lubricant used should be mineral oils with a high flash point and high decomposition temperature;

(c) compressors should be lubricated systematically, with the minimum quantity of lubricant;

(d) air receivers for compressors should be periodically cleaned, since they accumulate lubricant brought in as vapours by the compressed air and condensed in them.

1-3 6 Hydrogen, H_2

Hydrogen is a colourless and tasteless gas, physiologically inert, extremely light, with a specific gravity of about 0 09 referred to air, at ordinary mine temperatures and pressures it forms an explosive mixture in air at concentrations of about 4%, igniting more easily than methane, the most powerful explosion taking place at 28 6% hydrogen and 71.4% air.

In coal mines hydrogen is found usually in methane, together with heavy hydrocarbons, but it can be formed in mine fires together with other combustible gases from the gasification of coal.

Much more frequently hydrogen is found in potash mines, where it is often emitted with methane, and in German potash mines it has been found in "blowers".

The ignition temperature of hydrogen is 100-200°C below that of methane In particular, this must be considered in blasting and in the use of flame safety lamps, which are not safe in the presence of hydrogen even with two protective gauzes.

1-3.7 Ammonia, NH_3

Ammonia usually accumulates near stables, particularly if they are neglected, therefore the air flow from stables should always be directed immediately into the return airway.

1-3.8 Radiation by Radioactive Gases

Of the radioactive gases (emanations such as radon, thoron, etc.) radon which is encountered in uranium and thorium mines is particularly dangerous.

If radon and other radioactive gases are found in the mine air it becomes strongly radioactive—a hazard which is particularly severe because its damage to the human body is not immediate but is noticed only after some time

1-4. METHANE, CH_4

Methane or, as it is often known, *firedamp** is one of the most dangerous impurities in the mine air. It has repeatedly caused powerful explosions which have killed many men, and even in modern mining it is impossible always to prevent these disasters.

In the mines of the USSR methane was first discovered at the end of the last century, and ignitions of methane and air began in 1878 in some Donets Basin coal mines ** At present in the USSR more than 60 per cent of all coal mines are gassy, i.e. they emit methane

Impurities in methane The methane liberated underground almost always contains carbon dioxide and nitrogen, sometimes also hydrogen and heavy hydrocarbons (predominantly ethane C_2H_6, hydrogen sulphide H_2S, sulphur dioxide SO_2, and carbon monoxide CO).

The gas samples which are the richest in methane are those which are taken from cracks or from holes drilled in the coal; sometimes they contain up to 99.9 per cent of methane.

The carbon dioxide content usually does not exceed 5 per cent, the nitrogen content, a few per cent, but instances have occurred (most often in potash mines, rarer in coal mines) where the nitrogen content was 20 per cent or more.

Hydrogen, ethane, and ethylene are comparatively rare impurities, but they are undesirable because they are highly explosive and have both lower explosive limits and a lower ignition temperature than methane They also weaken an extremely important property of methane, to ignite not immediately but with some delay, a property described below. Hydrogen makes the methane flame (its cap) pale and barely visible. Usually in coal mines, hydrogen and the members of the methane series, considered as impurities in methane, do not exceed 1-4 per cent and only very rarely does

* By mine gases are usually meant the gases naturally emitted in mine workings, or formed as a result of chemical and biochemical reactions between the air and minerals or rocks containing them, timber supports, etc However, the frequently used term, *firedamp* (Russian *rudnichny gaz*, French. *grisou*; German *Schlagwetter*) is used for describing mine air which contains methane.

** Scientific research into gas and dust suppression underground is in progress now at many Soviet research organizations

The composition of mine gas in potash mines is particularly complicated since in addition to CH_4 and H_2 it contains nitrogen and sometimes hydrogen sulphide

methane contain them in large quantities. Thus, in Pennsylvania, USA, where in some places, the carboniferous series underlies petroleum-bearing beds, the methane sometimes contains up to 10 per cent or more of ethane, e.g. CH_4 82.4%; C_2H_2 16%; CO_2 0.1%; CO 1.5%.

The appearance of this sort of multi-component methane is possible also in the mines of Kizel (USSR) where bituminous ground underlies the coal-bearing rock.

Hydrogen sulphide and sulphur dioxide are quantitatively negligible but exceedingly unpleasant impurities in methane, partly because of their smell, and partly because of their irritating effect on the mucous membranes of the eyes and nose (especially SO_2).

Wheeler found that in methane samples taken from holes drilled in the coal in various British mines, there was from 0.5 to 1 per cent of carbon monoxide.

According to investigations by N. S. Kurnakov, M. N. Chernitsin and others, the possible contents of hydrogen and heavy hydrocarbons in methane in the mines of the Donets Basin are so insignificant, that they are within the limits of analytical error

A small quantity of hydrogen has been found in the Donets Basin in the rock containing coal seams, according to M. M. Elinson It has also apparently been found in the Kuznets Basin.

Properties of methane. Methane is a colourless, odourless gas. Its specific weight is 0.554 relative to air; at 0°C and 760 mm pressure, 1 m^3 of methane weighs 0 716 kg. Because of its lightness it accumulates very easily in the high places of workings and therefore special care is needed when working to the rise in gassy seams.* For the same reasons methane easily passes through porous partitions, because it diffuses 1.6 times as fast as air. Methane has no odour, but in some mines it is emitted with impurities such as hydrocarbons and traces of hydrogen sulphide which have a smell, these gases give the characteristic smell of apples to methane.

The effect of methane on the breathing resembles that of nitrogen. It is harmful only insofar as the oxygen content in the mixture with air becomes inadequate for breathing. We know that if air is diluted with nitrogen or a similar gas, man begins to suffocate only when the oxygen content falls below 12%. The quantity of methane (x volumes) which must be added to air so that instead of 21% there is 12% of oxygen can easily be obtained from the formula:

$$\frac{21}{100+x}=\frac{12}{100}$$

* After some time, if there is no new inflow of methane, the gas diffuses more or less uniformly throughout the working.

... which $x = 75$ volumes per 175 volumes of mixture, i.e the mixture will contain about 43% of methane A similar calculation shows that a fatally dangerous reduction of the oxygen content (to about 9%) will take place only with a methane content of 57% Nevertheless miners have died of methane suffocation in dead-end faces of vertical or steeply rising workings.

Methane is slightly soluble in water, at a pressure of 760 mm and a temperature of 20°C only 3.5 volumes of it dissolve in 100 volumes of water, and it is hard to liquefy; at +28°C a pressure of 250 atmospheres is needed to liquefy it.

Under ordinary conditions, that is, at low temperatures and pressures, methane is extremely inert chemically and combines only with the halogens, such as chlorine

Methane is flammable and forms explosive mixtures with air. When it burns or explodes it combines with the oxygen of the air to form carbon dioxide and water, the reaction giving the following initial volume relationships and end products*

$$CH_4 + 2O_2 = CO_2 + 2H_2O$$

$$1 \text{ volume} + 2 \text{ volumes} = 1 \text{ volume} + 2 \text{ volumes}$$

$$16 \text{ by wt} + 64 \text{ by wt} = 44 \text{ by wt} + 36 \text{ by wt}$$

The water vapour condenses and the volume of the combustion products of methane thus becomes one third of the volumes of the gases (methane and oxygen) which take part in the burning

If the reaction takes place not with pure oxygen but with air, the ratio of the reactants and the end products are as follows.

$$CH_4 + 2(O_2 + 4N_2) = CO_2 + 2H_2O + 8N_2$$

that is, one volume of methane is burnt by all the oxygen contained in 10 volumes of air From this it follows that the most powerful explosion is given by a mixture containing 1/11 of a volume of methane (about 9 5%) One kg of methane in burning evolves 13,300 kilocalories of heat, whereas 1 kg of gunpowder releases only 580 kcal and 1 kg of nitroglycerine gives out 1,500 kcal However in practice the gas explosion cannot be considered equivalent

* The reaction is shown in a simplified form and only indicates the end products of oxidation But it does in reality have a more complicated course, through a number of intermediate products, and proceeds by a chain mechanism This is extremely important for so-called catalytic flame-shielding By breaking the links between the intermediate products which are responsible for chain branching, that is, by breaking the chain, it is possible to retard the reaction and to prevent it from undergoing auto-acceleration to a high rate of gas ignition, or explosion This is the special role of [illegible] ... olders in permissible explosives They not only lower the [illegible] ... ature ... [illegible] ... tonation products of the explosive material but they ... id ... [illegible] ... hane by their catalytic, inhibiting properties

in intensity to an explosive because the methane being a gas cannot be ignited in an amount smaller than 1/11 of the space in which the explosion takes place, or in other words (at a pressure of 1 atmosphere and a temperature of 20°C) about 65 g per cu m of space, corresponding to a density of charge of 0 000062. For black powder the density is 1 and for nitroglycerine 1.6.

When the methane content in the air is more than 1/11 by volume, the oxygen content is inadequate for burning all the methane and the burning reaction can only take place partially with the appearance of end products including some quantity of hydrogen

$$CH_4 + O_2 = CO + H_2 + H_2O$$

A reaction between methane and oxygen which is subject to the laws of these calculations begins to proceed even at a temperature of about 300°C but the phase of the reaction which is accompanied by flame and is properly called ignition takes place only at much higher temperatures.

The ignition temperature of methane is usually taken as 650 to 750°C. However, the most recent investigations have shown that this temperature may be considerably higher or lower than these limits depending on the type of igniter, the method of ignition, the methane content of the air, the presence of various impurities in the methane, etc

For example, experiments have shown that hot surfaces ignite the gas at a temperature higher than a flame temperature, e g according to some data it is possible to conclude that for the iron gauze of a flame safety lamp to ignite methane, its temperature must be about 1200°C; on the other hand, methane ignites at 510°C if its mixture with air is adiabatically compressed to 60-70 atmospheres Other conditions being equal, the most flammable mixture of methane in air is a mixture containing 7-8% of methane.

The ignition temperature of methane depends both on the pressure of the mixture and on the temperature of the enclosing space it is lowered when the pressure and the temperature rise and vice versa.

The most characteristic property of methane is that, when in contact with a hot source, it ignites not immediately but with a delay, that is, after an interval of time which depends on the temperature of the hot source; at 650°C the delay is 10 seconds, at 1000°C it falls to 1 second The latest investigations have shown that the ignition of a 6.5% mixture of methane in a quartz electrical furnace takes place with a delay of 11 seconds at 700°C and only 0.02 second at 1200°C.

This is explained by the hypothesis that methane begins to dissociate and burn only after it has absorbed an appropriate amount of heat (22.1 kcal/mole).

At pressures above 1 atmosphere, the delay time also diminishes, but not so considerably as with an increase in temperature, and in mining conditions this has no practical significance.

The property of firedamp to ignite with a delay, however small the magnitude of this delay may be at high temperatures, is of great practical importance for safe shotfiring in gassy mines. Because the duration of the flame in the detonation of explosives can be reduced to a fraction of a millisecond, a methane-air mixture will not ignite under such conditions The theory of so-called permissible (for gas) explosives is partly based on this fact

The presence of hydrogen and other combustible gases in methane reduces the delay in ignition, e.g the presence of 30% hydrogen in methane eliminates the delay completely.

The colour of the methane flame varies with the conditions of burning from medium blue to very pale blue or even white. At very small contents of the gas in air, the flame is almost deep blue, at higher contents, e g. 5%, the flame becomes pale sky blue

The velocity of propagation of a methane flame depends on several factors Let us imagine a space filled with a mixture of the gas with air If within this space some part of the gas is heated to the ignition temperature, the molecules begin to burn and to heat the neighbouring molecules. If this heat brings them up to the ignition temperature, they also will burn The heat can generally spread by one of the two methods (a) by conduction and radiation, and (b) by detonation

A detonation occurs if the burning is so intensive that the neighbouring particles are compressed and heated up to the ignition temperature. The detonation method of flame propagation in gas mixtures underground is evidently not possible because hitherto detonation has only been observed when the gas is ignited in pure oxygen The greatest interest for us lies therefore in the dependence between the flame velocity and the general process of burning of methane on various factors for the case of flame propagation by conduction and radiation In this instance

1 The flame velocity depends on the methane content of the mine air The flame in general will not be propagated at a gas content less than 5 or 6%, nor at more than 14 or 16%, as the gas content passes from the lowest to the highest range indicated the flame velocity at first increases and then diminishes.

2. The important factor affecting the velocity of propagation of ignition is the condition of the methane-air mixture, that is, whether it is in movement or in a state of rest. The results of experiments on the ignition of methane in tubes have shown that if the mixture is at rest, the greatest velocity of propagation is observed at 10-12% of CH_4 and does not exceed 0 6 m/sec. If the mixture is in continuous

or fluctuating movement, the velocity rapidly increases up to hundreds of metres per second.

The flame velocity depends also on the place where it begins, e g if a tube sealed at one end is filled with a methane-air mixture and the gas is ingited at the open end, the flame velocity will not exceed 0.6 m/sec, but if it is ignited at the closed end the flame passes out many times faster.

3. The neighbourhood of cold bodies, which can absorb the heat, affects the flame velocity. If the particles of gas surrounding the burning part possess a high heat capacity, they will absorb all the heat produced in the burning reaction, and will not themselves be heated up to the ignition temperature, and the ignition will not be transmitted to neighbouring particles. This role is played by any cold body, e.g. the gauze of a miner's lamp. The gas heated within it passes through the openings of the gauze and loses part of its heat by contact with the gauze and cannot ignite the gas outside the lamp. Experiments on the ignition of gas in glass tubes of various diameters have shown that diameters larger than 50 mm have no effect on the flame velocity; at a diameter of 50 mm the flame velocity is 0 5 m/sec, at 8 mm diameter it is 0.39 m/sec, at 5 5 mm it is 0 22 m/sec, and at 3 2 mm it is almost zero; the flame in a tube of this latter diameter does not propagate at all or propagates only to a very small distance.

4. Constrictions, widenings, and any sort of obstacle such as solid bodies in the path of the gas compress the latter and bring it into a fluctuating condition which can convert the burning into an explosion.

5. The ignition and burning of methane depend on the composition of the air with which the gas is mixed. Either a lowered oxygen content or a high carbon dioxide content will make ignition or burning more difficult

Formerly it was believed that in an atmosphere with an oxygen content below 17.25% an explosion of methane or coal dust was impossible, and that to prevent explosions it was sufficient to maintain the oxygen content in the mine air around 17%, such air was considered to be physiologically harmless However, investigations by N. N. Chernitsin and others have refuted this dangerous conclusion and shown that the gas ceases to ignite only at oxygen contents below 12%.

The explosion temperature of a methane-air mixture, i e. the temperature to which the gaseous products of explosion are heated at the instant of explosion reaches 2150-2650°C, if the explosion takes place in a confined space (volume q = constant), and 1850°C if the products of explosion can expand quite freely (p = constant).

The highest pressure P is developed during an explosion with q = constant and in a mixture with 9.5% methane in which all the oxygen is burnt.

If the temperature and pressure of the methane-air mixture before the explosion are t and p, and after the explosion are t_1 and p_1 respectively, then evidently

$$p_1 = p\,\frac{273+t_1}{273+t} \tag{1-1}$$

from which, taking t = 15°C, and t_1 = 1850° or 2650°C, it follows that p_1 = 7.4 to 10.1 p, that is, p_1 is approximately equal to 7 to 10 p

Thus, the greatest pressure which can develop in the explosion of a methane-air mixture is approximately 9 times as much as the pressure before the explosion This means that if p = 1 atm, then p_1 = 9 atm In determining this pressure in laboratory experiments, it was observed that the maximum value of p was 9.05 atmospheres at a methane content of 10% methane. It would, however, be a mistake to believe that in gas explosions underground no pressure greater than 9 atm can be obtained.

In reality, during the propagation of the explosion wave along the working, a wave of compressed air precedes the flame This wave, meeting accumulations of gas, compresses them and the approaching flame ignites them, the pressure developed by the second explosion being therefore equal to P = 9 p, where p is greater than 1; P can thus be considerably greater than 9 atm From this it follows that the places in the mine where the greatest mechanical damage and destruction are caused by an explosion are not necessarily the sources of explosion, on the contrary, the source of the explosion usually suffers less damage than the locations of the secondary explosions

Flammability and explosibility are the most important properties of methane as far as mines are concerned As pointed out above, these properties vary, depending on the percentage content of methane in the mixture A distinction should therefore be made between the mixtures containing

(1) up to 5-6% CH_4,
(2) from 5-6 up to 14-16% CH_4,
(3) above 14-16% CH_4

The first mixture is not explosive, but it can burn near a hot source The second mixture is explosive. The third mixture is not explosive and does not support combustion, but when oxygen is added to it from outside it burns with a tranquil flame like lighting gas

The most easily ignited mixture is the 8% mixture, but the most powerful explosion takes place at a methane content in air of 9.5%

because in this case all the oxygen is burnt. As indicated in Table 1-7, at gas contents below 9 5% part of the oxygen remains unburnt, and at contents above 9.5%, since there is not enough oxygen in the air, only part of the methane burns and the remainder cools the mixture; in addition, the volume of the gases taking part in the reaction is less than with a 9.5% mixture.

TABLE 1-7 **Combustion of Oxygen in Relation to the Methane Content of the Air**

Enough CH_4 is added to normal air to give the mixture		For complete combustion of the CH_4, the O_2 percentage needed is	Oxygen, per cent		Volume of the gases participating in the reaction, per cent
CH_4, per cent	O_2, per cent		excess	deficiency	
9 0	19.1	18.0	1 1	—	27 0
9.5	19.0	19.0	—	—	28.5
10.0	18 9	20 0	—	1 1	28 3
14.0	18 0	28.0	—	10 0	27 0
16 0	17.7	32.0	—	14.3	26 6

At a methane content above 14-16% in air the mixture ceases to be explosive. This is explained by the fact that methane has a high specific heat—0 59 (compared with 0.22 for oxygen, 0.24 for nitrogen, and 0 23 for air) and also by the fact that the heat emitted when burning begins is absorbed by the surplus gas which splits into acetylene, C_2H_2, and hydrogen, H_2

These are the explosion limits of methane-air mixtures However, if the air contains not pure methane but a mixture of combustible gases (methane, ethane, hydrogen, etc), then these limits of explosibility will be quite different

Extremely careful laboratory experiments on the conditions of change of the explosion limits of mixtures of mine gases have brought out the following conclusions

1. The lower limit (x%) of explosibility of a mixture of methane and other hydrocarbons liberated underground with normal air, for usual mine temperatures and pressures, can be calculated from the formula of Le Chatelier.

$$x=\left[\frac{100}{\frac{P_1}{N_1}+\frac{P_2}{N_2}+\frac{P_3}{N_3}+\cdots}\right] \text{ per cent}$$

where P_1, P_2, P_3, = contents, in per cent by volume, of each combustible component of the mixture ($P_1 + P_2 + P_3 + \ldots = 100\%$)

N_1, N_2, N_3, ... = lower explosion limits of each component *

Thus, for example, if in the mine the gas emitted is a mixture of 80% methane, 10% hydrogen, 10% ethane, then the lower limit of explosibility of this gas mixture will be.

$$x = \frac{100}{\frac{80}{5} + \frac{10}{4.1} + \frac{10}{3.2}} = 4.6\% \text{ approx.}$$

2. The explosion limits of a mixture of methane with air containing various percentages of oxygen are presented in Fig. 1-8 which shows what mixtures of methane with air are not explosive at a given content of oxygen, and with what oxygen contents a given methane-air mixture remains explosive For example, at oxygen contents of 18% and 17% the explosion limits of a mixture of methane with air are 5.3-14.0% CH_4 and 5.3-12.6% CH_4, respectively, a mixture containing 10% CH_4 ceases to be explosive at an oxygen content below 15%.

Methane explosions are always accompanied by two shocks: the *forward shock* caused by the heated gaseous products of the explosion under high pressure creating an air wave of a considerable force travelling rapidly away from the point of explosion, and the *second* or *reverse wave* which results from the pressure drop at the point of explosion because of the cooling of the gases and the condensation of the water vapour (occupying a considerable part of the volume of the explosion products).

Generally speaking the reverse wave is of a rather smaller force than the primary wave, but since it follows the course just traversed by the first wave, which has done considerable damage, the mechanical effects of the reverse wave are very often much more stronger

At gas contents in the mixture higher than 9.5% there can be two types of flame the primary flame spreading with a high velocity and burning all the oxygen of the air, and the secondary flame moving backwards more slowly, since all the oxygen has been burnt, this latter flame is caused by the combustion of the remaining gas by the oxygen which has moved into the explosion area from outside

After the explosion, the explosion area is filled with a mixture of hot gases, impoverished in oxygen or nearly devoid of it, consist-

* The explosion limits of mixtures with air of normal composition for ordinary mine temperatures and pressures are methane 5.0-15%, carbon monoxide 12.5-74%, hydrogen 4.1-74%, acetylene 3%, ethane 3.2-12.5%, petrol 1.1-5.8%.

ing mainly of nitrogen and carbon dioxide, and sometimes containing small amounts of carbon monoxide.

Carbon monoxide is formed in large quantities in gas explosions involving coal dust, when the carbon monoxide content may reach several per cent

The gases resulting from explosions spread into the neighbouring workings and make the atmosphere poisonous and unbreathable. They are the main reasons for the deaths of the miners who are

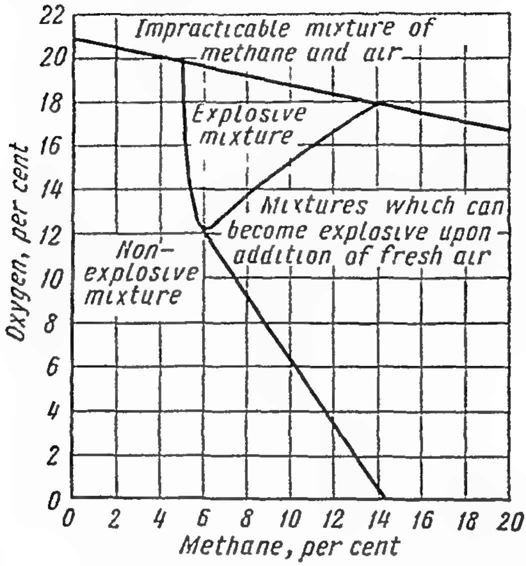

Fig 1-8. Limits of explosibility of a methane-air mixture

underground during the explosion. Investigations of the consequences of underground explosions have shown that usually not less than two thirds of the fatalities result from carbon monoxide poisoning or from asphyxiation, which is oxygen starvation, with an accumulation of excess carbon dioxide in the body of a man or an animal.

Origin, location and processes of conservation of methane. If the remains of dead vegetation are protected from the oxygen of the air, they begin to undergo the process of *peat formation* natural to organic substances; the vegetation loses its structure and is converted into a complex brownish-black substance which eventually becomes coal. The essence of this process is the slow oxidation of the vegetable remains by the oxygen contained in the vegetation itself; the final gaseous products of the process are carbon dioxide, methane, water vapour, volatile organic acids, etc. Under certain conditions the reaction produces mainly methane and carbon dioxide (the so-

called *methane fermentation*); the cellulose ($C_6H_{10}O_5$) decomposes with the following end products.

$$2C_6H_{10}O_5 = 5CH_4 + 5CO_2 + 2C$$

or

$$4C_6H_{10}O_5 = 7CH_4 + 8CO_2 + 3H_2O + C_9H_6O$$

in which C_9H_6O is the solid remnant (coal).

The process of gas formation continues, though less intensively, during the later period of metamorphism of the coal.

It has been established that the stimulants for methane fermentation are special micro-organisms bacteria (*micrococcus carbo*), and bacilli (*bacilla carbo*) The process of methane formation depends on the composition of the fermenting substances and also on the temperature and pressure

If the gases formed in the peat formation of the vegetable remains can be released, they disappear with no trace left If not, specially if the decomposing vegetable remains are covered, say, with a layer of sand, silt, or other ground, the gases remain in their place of formation in the solid remains of decomposed vegetation or in the surrounding rock

It follows from this that the methane, pure or mixed with other gases, can be encountered in all those places where the methane fermentation of vegetable remains has taken place and where the resulting gases were not released

In reality, methane is emitted not only in hard-coal, anthracite, and brown-coal mines, but is also found in rock-salt and particularly in potash mines, as well as in strontianite and sulphur mines, in clay pits, and also in individual iron mines (the hematite mines of Cumberland, England), in lead and other mines For example, in the complex metal mines of Norilsk (USSR), where in the footwall of the ore deposit there is a succession of sedimentary rocks (carbonaceous shales and coal seams), methane is emitted through cracks from the rock into the workings in the form of "blowers' which sometimes last for years

Methane is often emitted from the gold placers of Western Siberia, of the Far East (Kolyma) and in the Urals (Shural), the rocks lying under the apatite deposits of the Kola Peninsula also contain methane In the South-African gold mines methane is emitted simultaneously with hydrogen (up to 20-30%)

In the clay mines near Borovich in the Leningrad region, gas ignitions repeatedly occurred when naked lights were used

Particularly well known are methane emissions which occurred at the clay mines of Malakov near Paris, also near Namur in Belgium, and at Loten in Saxony, where explosions have caused human deaths. The presence of the gas in these clay mines is explained partly by

the fact that the clay contains up to 5% of organic material, and partly by the decomposition of timber supports left in old workings.

In the mines of the Moscow region several methane ignitions have occurred from naked lights; in these mines the methane was formed by fermentation in sumps.

Large quantities of methane are most frequently found in coal mines, and therefore we shall mainly consider coal mines.

The methane which is released into the mine openings as the confining pressure is reduced occurs in the mineral (coal) deposit itself or in adjacent or nearby formations either in the form of a "*free*" *gas* (in the pores, cracks, and voids in the coal seams and adjacent rocks) or as a "*combined*" *gas*, of which two types exist the *adsorbed gas*, i.e that which is concentrated on the surface of the porous mass of coal or carbonaceous rock (the forces binding the adsorbed layer to the surface are physical in nature), and the *absorbed gas*, i e. the gas which has diffused into the interior of the coal, forming with the latter a sort of solid solution

The adsorbed gas is described as "combined" to differentiate it from the "free" gas. The total surface area to which the molecules of the gas adhere is very large and reaches 100-200 million m² per ton (or an area of 10 × 10 to 14 × 14 sq km). Many of the pores are completely isolated from each other, they contain 20-30% or up to 40-50% of all the methane. The gas is not released from these pores even when the coal is heated to 150°C. With increased pressure on the coal, or, which amounts to the same thing, with increased depth, the volume of the free gas increases, and that of the adsorbed gas (combined) diminishes, as can be seen from Table 1-8 (according to Lidin).

TABLE 1-8 **Distribution of "Free" and "Combined" Methane in Coal**

Depth, m	Pressure, atm	Temperature, °C	"Free" gas		"Combined" gas		Total, m³/ton
			m³/ton	%	m³/ton	%	
50	5	8.3	0.8	8 7	8.4	91 3	9 2
100	10	9.9	1 5	11 4	11.6	88.6	13 1
250	25	14.5	4 2	23 3	13 3	76 7	17 5
500	50	22 2	8.2	39 2	12 7	60.8	20.9
1,000	100	37.6	15 5	57.9	11.7	42 1	27 2
2,000	200	68 5	25 0	77 4	7 3	22.6	32.3

If the pressure is removed from the coal, the gas moves over from the combined state to the free state. This fact plays an important part in so-called sudden outbursts which are discussed below.

Apart from the gas physically combined with the coal, there are chemically combined gases which are liberated on heating—methane, carbon dioxide, hydrogen and others (so-called volatile matter).

Part of the gas is also contained in the surrounding rock (see below).

The three typical kinds of gas emission underground from primary sources (the coal seam itself or the adjacent rock) are as follows

1 Ordinary emission from pores and cracks invisible to the eye in the coal or rock throughout their exposed surface—a slow but continuous and prolonged emission.

2 Blowers from visible cracks and other openings in the coal or rock, sometimes short-lived but more often prolonged, up to 2 years or more

3 Sudden outbursts of methane or carbon dioxide or both together, sometimes in a very large quantity from the coal seam or the rock, and accompanied by the ejection of large quantities of finely divided coal and therefore usually called sudden outbursts of coal and gas

These three types of gas emission are considered below.

1-5. ORDINARY EMISSION OF GAS

Coal Structure. For the proper understanding of the process of gas emission it is necessary to know the structure of coal which is the main source of gas in mines.

Coal is composed of (a) crystallites, measuring 15-30 Å; and (b) micelles, consisting of groups of crystallites, the dimensions of micelles are 10-2000 Å, or 0.000001-0.0002 mm.

Within the crystallites there are ultrapores measuring up to 100 Å, within the micelles—micropores measuring millimicrons; these pores can be seen in the electron microscope.

Note 1 Ångstrom (Å) $= 1 \times 10^{-7}$ mm, 1 millimicron $= 1 \times 10^{-6}$ mm, 1 micron $= 1 \times 10^{-3}$ mm

The porosity of long-flame coals averages 17% (10-22%) and of other coals 8-10% (1 3-26%). The porosity of the surrounding rock averages 5-8% (0-28 7%).

The pore surface consists mainly of the finest pores, which make up 97% of the pores With increase in the volume of the pores, the volume occupied by any particular pore size increases but the pore surface diminishes and vice versa The smallest pore volume is found in coals containing about 20% volatile matter.

Underground observations and laboratory and theoretical work have led to the following conclusions:

(a) At some distance ahead of the coal face within the seam, the effect of the abutment pressure (which may be 1 5 times or twice as large as the average pressure of the overlying ground) gives rise to a zone where the pressure is increased and the porosity of the coal is at a minimum; this zone can be called the *threshold of consolidation* This threshold is a sort of barrier which restricts the gas emission from the coal mass into the worked-out area the pores and cracks are compressed under pressure and do not release the gas

(b) The distance from the edge of the seam to the threshold of consolidation increases with the thickness of the seam and with the softness of the coal.

(c) In the coal between the coal face and the threshold of consolidation, the porosity of the coal considerably increases because of release of rock pressure and the breaking of the coal

Permeability of Coal to Gas. Gases can penetrate the coal mass because of the porosity of coal. The permeability in the virgin coal is considerably lower than that in the zone of the face. The permeability rapidly diminishes with depth into the coal. The minimum permeability is found in anthracite coals The permeability parallel to the bedding planes is considerably higher than that perpendicular to them, and in rock it is considerably less than in coal.

The Paths of Gas Migration. In coal mines methane travels from one place to another by various processes.

1 The gas emitted from the coal itself or adjacent formations into the mine passes rapidly along with the ventilating current When much gas is emitted, it forms a layer at the roof and the layer travels with the air current over a distance of some tens of metres, until it mixes fully with the air; this danger is discussed below. After the gas has mixed with the air it does not separate out again

The gas moves with the air stream because of the difference in the air pressure created by the work of the fan Outside the productive workings, the gas also travels through the worked-out areas and in the large hollow spaces formed in the roof rocks when they cave or separate.

2 The second process of gas migration is by filtration through the numerous cracks in the coal and surrounding rock; this type of movement is characteristic of the face area (see below) The movement of the gas is caused by the same pressure difference as in (1).

3. The third process of gas migration is by diffusion This is migration of the molecules of a gas through the molecules of another substance; for the gas to move by diffusion there is no need

for a pressure difference, diffusion takes place by the motion of the molecules themselves, the number of molecules moving from a point with a high concentration to a point with a low concentration being greater than that moving in the reverse direction.

Usually both processes—the movement of gas by pressure difference, i.e by air flow, and that by diffusion—take place simultaneously, the first process predominating.

Examples of diffusion of gases are

(a) the gradual distribution of the gas liberated in dead-end roadways, the molecules of the gas diffuse through the molecules of air,

(b) the gradual dilution of the gas which accumulates in hollows in the roof,

(c) the movement of the gas from the worked-out areas into the face if the gas content in the former is higher than in the latter.

As a result of the movement of gas from the coal mass towards the face both by filtration and by diffusion, its pressure falls and the so-called drainage area is formed near the face The size of this zone is determined by the original gas pressure and the gas permeability of the seam. For the Vorkuta, for example, at Mine No. 40, it is equal to 25 metres, at Mine No 18, it is 24 m, and at Mine No. 25, it is 12 m.

The last method of movement of gas underground is together with mine water, on the other hand, water which floods a coal seam near its outcrop will seal up the gases by blocking the small pores

Gas Content. The gas content of a seam is the quantity of gas per unit weight of virgin coal under natural conditions. It is expressed in m^3/ton and depends on.

(a) the density of coal in the strata, i e the ratio of the thickness of the coal seams to the thickness of the surrounding rock,

(b) the degree of metamorphism of the coal, the more the coal has been metamorphosed the more gassy it is liable to be,

(c) the duration of the denudation cycles, that is, the time for which the coal measures have been exposed and consequently the gas has been able to migrate to the surface,

(d) the geological structure of the coalfield, which either facilitates the release of gas or makes it difficult, for example, if part of the deposit is folded, it is likely to yield its gas easily,

(e) the presence and the type of tectonic disturbances.

The maximum gas content at the depths now being worked does not generally exceed 30-40 m^3/ton, the gas content of very porous coals or very fissured coals may reach 50-60 m^3/ton

The gas yield from the rock may be a fraction of that from the coal However, if we remember that the thickness of rocks is many times the thickness of the coal seams, the total quantity of gas contained in the rock may be more than that in the coal seams.

Methods of Determining the Gas Content. At present the following three basic methods are used for determining the gas content:

(a) the direct determination of gas content by various coring methods; the core brought out by drilling is measured in the laboratory for its gas content;

(b) so-called indirect determination by measurement of the gas pressure in the face of drilled holes with subsequent determination of the gas content of the coal in the laboratory by pressure measurement;

(c) so-called gas coring—the determination of the gas content by the quantity of dissolved methane in the drilling fluid and the amount of the gas remaining in the coal cores

All three methods of determining the gas content give low results

On the basis of data on gas content, maps can be drawn, showing contours of equal gas content.

Gas Emission. There are two types of gas emission. (a) the *absolute* or *total gas emission*—the total volume of gas emitted from any mine during 24 hours, in m^3, and (b) the *relative gas emission*—the quantity of gas per ton of daily output.

The absolute gas emission in very gassy mines with a high output can be extremely intensive, reaching several hundred thousand m^3 of gas per day.

The relative gas emission depends on the presence of accompanying seams near those being mined; if these seams are present, the gas emission increases because of the additional emissions from them and, partly, from the adjacent rock. In the USSR some mines have 100 or more m^3/ton, and in other countries the value reaches 250-300 m^3/ton In the USSR all coal mines are classified according to their gas emission in one of the following categories.

Category 1—less than 5 m^3/ton
Category 2—from 5 to 10 m^3/ton
Category 3—from 10 to 15 m^3/ton

Mines with more than 15 m^3/ton of daily output are considered to be outside the categories

According to current Soviet Safety Regulations, mines working seams which are subject to sudden outbursts of coal and gas and to blowers are also considered outside the categories (see below).

Under Soviet Safety Regulations the gas emissions of all coal mines are measured annually. In June and July on days of normal mine output at the beginning, middle and the end of the month air samples are taken from the return airways and the air flows are measured simultaneously; the observations are made three times a day. The average gas emission for each day is determined; out of

the three determinations the maximum is taken, which is divided by the corresponding tonnage output

Forecasts of Gas Emission. The gas emission forecast for a working mine when a new horizon is opened is different from that for a mine being planned For a working mine the so-called mining statistical method is used, it consists of plotting on a diagram the known gas emissions of the various horizons and joining the points thus obtained, extrapolation then gives the gas emission at deeper horizons, extrapolation is allowed for depths less than 100 metres in mines of categories 1 and 2 and for depths less than 200 metres in mines of higher categories

The second method is much more complicated, special instructions for forecasting gas emissions by using the gas content data give formulas which are employed for determination of the quantities of gas expected from the various sources, these are added and divided by the planned daily tonnage

Rate of Increase of Methane Emission. The rate of increase of methane emission is expressed by the vertical depth in metres by which the gas emission increases by 1 m³/ton If the rate of increase is constant, the increase of gas emission follows a linear law according to the formula

$$q = \frac{q_0 (H - H_0)}{a} \tag{1-2}$$

where a = rate of increase of methane emission
H and H_0 = depth at which the gas yield is determined, and depth of the zone at which gas emission begins (the depth of the weathered zone), metres
q_0 = gas emission per ton at depth H_0

It is believed that at great depths, exceeding 1000-1200 metres, the rate of increase of methane emission diminishes The rate of increase of methane emission in the main coalfields of the USSR are as follows. in the Donets Basin coal mines usually 20-25 up to 35-40 metres, but in the district of Khrustal it is 4-5 metres, in the Kuznets Basin coal mines it is 10-15 and up to 20-25 metres, in Karaganda it is 10-15 metres but may be 6-7 metres

Sources of Gas Underground. Gas is liberated

(a) from the coal face—the exposed parts of the seam in development openings,

(b) from the broken coal in the face and during transport, also in bunkers on the surface (gas explosions have taken place in them);

(c) from nearby seams and unworkably thin seams;

(d) from the enclosing rock,

(e) from pillars of coal or areas which have been left.

From sources (c), (d) and (e) the gas is released mainly into the worked-out area (waste) and comparatively little into the face area; the gas emission is very much affected by the overworking and particularly by the underworking of seams. With increasing depth of working the waste becomes increasingly important as a source of gas.

The Gas Balance of the Mine. The gas balance of a mine is the distribution of the methane emitted from various sources, it is usually expressed in per cent of the total gas emission from the mine (or district). For example, the average gas balance of a mine in the Donets Basin is given below

from the exposed surfaces of the seams being worked	40-50%
from broken coal	10-15%
from the rock into the face	10-15%
from the rock into the waste	20-25%
from unworked coal seams	
into the face	0-5%
into the waste	5-10%

With the rapid increase in the rate of development work, the gas emission in development openings can reach 40-50% or even 70%; if there are nearby gassy seams the gas emission in the waste can reach 80%.

Factors Affecting Gas Emission. The factors affecting gas emission are very numerous, the most important of them being.

(a) the method of mining and the type of coal-winning machines;
(b) change in the level of output,
(c) change in the speed of advance of the face or of the general production front;
(d) the ventilation conditions.

The gas emission in the face is not uniform; it increases abruptly when cutter-loaders are at work and greatly diminishes in a repairing shift. The variation factor of the gas emission is the ratio of the average emission during the working shift to the 24-hour average emission: it depends on the duration of the work of the cutter-loaders; the greater the duration, the smaller is the variation factor; the variation factor of the production faces is greater than that of the districts, and the districts have larger variation factors than the mine as a whole. This is explained by the fact that in the producing districts, part of the gas emission is not associated with the coal-winning machines, and in a mine with several producing districts their working cycles do not coincide.

For the conditions of the Donets Basin coal mines, A. S. Tsirulnikov gives the following values of the variation factor:

In the face				In the district			
duration of work				duration of work			
6-8 hours		12 hours		6-8 hours		12 hours	
coal-cutter	cutter-loader	coal-cutter	cutter-loader	coal-cutter	cutter-loader	coal-cutter	cutter-loader
1 3-2 0	1 5-2 5	1 2-1 5	1 3-1 5	1 3-1.7	1 4-2 0	1 1-1 3	1 2-1 4

This table shows that the variation factor with cutter-loaders is rather higher than with coal-cutting machines alone

In conditions other than those in the Donets Basin coal mines where thin seams are worked, the variation factor will naturally be different.

The methane content within the cut in the coal reaches 50-80% at a depth of 1 metre. The methane content in the air flow past the coalcutter often exceeds the permissible value (1 per cent) and at the cutting, motor and drive sections, often reaches 5% which is an explosive concentration But since coal cutting is always dusty, an explosion is possible at an even lower gas content than this. A mixture of methane and coal dust will explode at

3% gas with dust amounting to 5 g/m^3
2% gas with dust amounting to 10 g/m^3

The methane liberated during the work of cutter-loaders quickly mixes with the ventilating air stream and usually at 4-6 m away from the machine its content does not exceed 1-1.5% The methane concentration rapidly increases when the undercut coal drops. When a coal-winning machine stops for 2 or 3 minutes, the gas content drops by several per cent; after the second start of the machine the gas emission rises to a maximum again after 5-10 minutes, it is possible therefore by periodically starting and stopping a machine to keep the gas content down to the permissible value, with the machine working only 30-50% of the whole shift

The size of the broken coal has a great effect on the amount of the gas emission in the face, in the first ten minutes only 2% of their total gas content is released from the large lumps, but from the fraction of a size of 0 25-1 mm, 40% of the gas is released, and from the smallest coal (below 0 25 mm) 66% of total gas content is released.

The main part of the gas is released, however, not from the broken coal, but from the freshly exposed surface of the coal seam (up to 50-75% of the total gas emission in the coal face). It is true that

it fairly rapidly diminishes. in the first 1 or 2 hours after exposure the emission from the surface is about 75-83 per cent.

It must be remembered that the coal face can also receive gas out of the waste, if the conditions are appropriate for much gas to be liberated from the waste (in the presence of nearby seams and in the case of working by caving) and with a direction of the air flow out of the waste towards the face (which depends on the method of ventilation, see below) the amount of gas released from the waste may be considerable and, as explained above, may reach or exceed 50 per cent.

Apart from the factors above, which affect the gas emission in the face, the absolute value of emission depends naturally on

(a) the gas content of the coal at depth and in the face being worked;

(b) the permeability to gas of the coal, the larger this is, the more easily will the gas migrate out of the virgin coal towards the face area.

The non-uniformity of gas emission during the work of coal-loaders was mentioned above, it causes the amount of air flowing through the face during the work of the machines to be increased by 50% or more, which leads to severe difficulties. Below are listed some measures which can help to reduce the gas emission during the work of cutter-loaders

(a) reduction of the thickness of the coal-cutting kerf by the use of a thin jib;

(b) cutting in a coal band which contains less gas, or overcutting the coal,

(c) increasing the size of lump coal

—by the use of machines for so-called coarse chipping;

—by appropriate layout of the cutting picks on the chain;

(d) reduction of the depth of the cut by shortening the jib

Very great reductions of gas emission (and dust formation) can be achieved by the use of Cardox, Hydrox, and Airdox for breaking coal.

The methods recommended for lowering the gas emission are particularly important in very gassy mines, where the ventilation does often fail to handle large quantities of gas, particularly since the quantity of air passing through the face is limited to a maximum velocity of 4 m/sec under USSR Safety Regulations. With manless mining, the velocity is not limited. To increase the safety of work in these conditions the following are recommended.

(a) the use of hurdles in the face to direct the air flow towards the cutter-loader; and the face should be cut downhill;

(b) systematic gas measurements or the use of automatic gas analysers for stopping work in good time before the gas content reaches the permissible limit,

The ventilating layout of a district has a considerable effect on its gas balance. There are four basic types of ventilation layout: with advancing longwall the ventilation can be either reversed or forward, and with retreating faces the same two possibilities exist.

Layout 1, advancing faces with reversed ventilating current, has the largest amount of leakage through the wastes and, other conditions being equal, the smallest quantity of air reaching the face. Passing through the wastes the leakage air pushes out the gas accumulated in them and carries it into the return airway; from this point of view the leakages can be regarded as useful; the other positive aspect of the method is that it provides the smallest quantity of air to the face, when the gas in the face is adequately diluted, this facilitates the ventilation of the district. it reduces the loss of ventilating pressure in the face, reducing the air flow velocity in it and the dustiness of the air in the face; and because the velocity of air flow in the face (maximum 4 m/sec) limits its length, it is true to say that air leakage through the wastes permits the face length to be somewhat increased.

But losses, even if useful, must not be excessive and they can be considered normal if they dilute the gas they sweep out of the wastes down to the permissible level provided that the gas emitted in the face is also diluted down to the permissible level

Layout 2 is the reversed flow of ventilation with a retreating face. The losses of air approaching the face are zero and consequently the face receives all the air arriving in the district, the waste is ventilated less effectively with this method than with the first, but it does receive some air and the proportion of gas swept out of the waste into the return airway can reach 40% or more of the gas balance, not counting the gas which is driven out of the waste into the face

In this method of ventilation, the most dangerous place with the highest methane content is the junction between the face and the return airway.

Layout 3 is with an advancing face and forward ventilation flow. Unlike the first method, the gas pushed out of the waste joins the return airway at practically the same point as the air from the face and creates ventilation difficulties, the quantity of air in the face continually increases as it approaches its outlet

Layout 4 is the use of a generally retreating face with a forward ventilating flow. All the air reaching the district is supplied to the face but it immediately begins to enter the waste and consequently the amount leaving the top of the face is less than that entering at the bottom, most of the gas emitted in the waste reaches the return airway without entering the face. If much gas is emitted during the work of the cutter-loaders, the air at the upper end of the face is sometimes inadequate to dilute the gas down to the permitted level.

Generally we can say that from the viewpoint of efficient face ventilation, the second layout is the best, after which come the third and fourth, a preference for one or the other layout depends on the gas balance of the district The smallest quantity of air reaches the face with the first layout, which, however, is best from the point of view of removal of gas from the waste The highest gas concentrations are found with the third layout at the outlet from the face and with the second layout at the junction of the face with the return airway

It remains to investigate the effect of a change of air flow into the face Only recently the basic method of reducing the gas content in any face was to increase the air flow into the face. It was considered that the methane content was inversely proportional to the quantity of air reaching the face where the methane was emitted This is not so Numerous observations underground have proved the following:

(a) at some mines the methane content diminishes with increase of the air flow to the face, as was formerly believed;

(b) at other mines increase of air flow to the face does not reduce the methane content in the return air and the amount of gas increases;

(c) finally in the third type of mines there is at first a rapid increase, in the face as well as in the district, in the quantity of gas released and in its percentage content, with an increase of air flow to the face. After this so-called "splash", the increased content and total amount of gas are maintained for some time but then they slowly diminish, sometimes down to the original level; in other mines they remain rather above the original level

With reduction of air flow, in a number of instances a reduction of the methane content has been noted, even including a reduction of the percentage content

To explain these contradictory consequences of an increase of the air flow the following should be taken into account: almost every coal-producing district is a complex of the coal face combined with the waste which possesses the following characteristic features:

(a) They form a parallel (split) ventilation system

(b) The cross-sectional area of the coal face is more or less constant and does not change with a change in the air flow, the area of the waste through which the air moves, while it passes in depth through the waste, increases with an increase in the air flow to the district as a whole

(c) As a consequence of (b) the width of the air flow passing through the waste increases and begins to include deeper zones which did not previously take part in the air flow and gas exchange, since these zones generally contain high concentrations of gas, the result of their inclusion in the general air flow and gas exchange is an increased

gas content of the air. This leads
case the waste zone ventilated by
level and with increase of the air
increase only slightly; in the other
tinue to increase with increase of
first instance the quantity of gas mecha-
increases at all, in the second be limit-
occurs.

Underground observations accompa-
should be increased slowly ve) and the
to reduce their effects. gas emission

The relation between the involves the
and the gas emission, and also e seams to be
important in connection
action should be taken entilation is the
part of the mine? If the ader Soviet Safety
lead to the undesirable content depends on

Methane Emission ace (also on the gas
tion of ventilation amount of air passing
is not the automation e that the gas emission
mine fan and the o the face is also uneven
tion underground. stance of districts in the
USSR more than ncluding coal-winning and
plants are now hange by a factor from 1 5

Automation of m 2 to 9 and for the junctions
by a methane or of 4. But since the change of
districts is autom ortional to the square root of
sidered solved. e. the amount of air reaching
first hat reaching the face can be

(a) fully
developed; ocess of dilution and removal of

(b) means mplicated; it is the result of two
pickups and low, which alter in time and are
lation will interaction results in the air con-

(c) the m the face
ed; years ago, first of all in Britain,

(d) stable layers of methane occurring
tions" to of metres Examination of the evi-
ing the sions showed that they were caused

(e) s

tion of a stable layer of methane
eration rate, the velocity of air flow
king, as well as on other secondary
ether in any given roadway a stable
d. various indexes, or layering num-
xample, use may be made of the expres-

operator will redistribute the air between the districts. Further, in the second stage of automation, the information will be sent to a computer in which the ventilation layout of the mine will be stored. On receipt of information about unfavourable ventilation in some district of the mine the computer solves the problem for the most favourable redistribution of the air under the given conditions, after which it sends the appropriate signal to the devices for regulating the air flow

The task is complicated by the indeterminate relationship mentioned above between gas emission and total air flow in a district.

Without delaying with the details of automation let us consider for a moment the location of the pickups. If the methane pickup is installed on the coal-winning machine itself, whenever the gas content reaches 2% (the maximum content allowed by the USSR Safety Regulations) two automatic devices will operate, one of these switches off the power from the machine and the machine will stop, the other automatic device sends a signal to the mine control room from which an order goes out to redistribute the air flow in the mine After several minutes the gas content will drop down to the permissible level and both automatic devices will function again This arrangement is obviously impracticable. Therefore the methane pickup should be installed either above the face or somewhere in the return airway where the methane content is more or less stable. Considering the ventilation layout this point could be either 100-150 metres away (the first of the ventilation layouts considered above) or directly in the face (in the second ventilation layout). It is also possible to use delay-action pickups

The Effect of Full Mechanization on Gas Emission. The introduction nowadays in Soviet coal mines of all-round mechanization must have a large effect on gas emission The speed of driving of development workings will reach 500-1500 metres per month in the coal seam and 300-500 metres in rock; the monthly advance of a face 150-200 metres long in a gently dipping seam about 1-1 5 metres thick using cutter-loaders will reach 80-100 m per month, that is, 3-4 m per day, although the output of the machines is not yet fully achieved even with these speeds, and a daily output from one face can reach 600-1,000 tons

This sort of intensified work naturally leads to a considerable increase in the gas emission. But because full mechanization will be mainly introduced in new deep mines we must expect increased gas emissions up to 40 m^3/ton in anthracite mines and up to 22 m^3/ton in mines working coking coal. From this gas emission, with a variation factor of 1.5 we obtain the following possible daily advances of coal faces in metres

Face length, metres	Anthracite	Coking coals
100	2 1	3 2
150	1 4	2 2
200	1 0	1.6

These face advances are less than what is possible with full mechanization (see above) because the work of the machines will be limited by the ventilation Therefore its introduction should be accompanied by measures to reduce the variation factor (see above) and the gas emission. One measure which may diminish the gas emission in a district by 50% or more and in the mine by 25-35% involves the preliminary degasification (firedamp drainage) of the seams to be worked (see below).

The main indication of the efficiency of the ventilation is the methane content in the return airway which under Soviet Safety Regulations must not exceed 1 per cent. This content depends on the absolute quantity of gas emitted in the face (also on the gas reaching the face from the waste) and on the amount of air passing through the face. It was pointed out above that the gas emission in the face is not uniform; the air flow into the face is also uneven. Investigations into the aerodynamic resistance of districts in the Donets Basin coal mines with a cycle including coal-winning and roof control have shown that it can change by a factor from 1.5 to 3 8, that of the faces by a factor from 2 to 9 and for the junctions of the face with the airways by a factor of 4 But since the change of the air quantity is inversely proportional to the square root of the resistance, then during a cycle, the amount of air reaching the district can be doubled and that reaching the face can be tripled

It therefore follows that the process of dilution and removal of gas from the face is extremely complicated; it is the result of two processes—gas emission and air flow, which alter in time and are independent of each other, their interaction results in the air containing the gas at the outlet from the face.

Layering of Methane. Some years ago, first of all in Britain, attention was turned to the fairly stable layers of methane occurring at the roof level for some tens of metres. Examination of the evidence of a number of gas explosions showed that they were caused by the ignition of these layers.

The likelihood of the formation of a stable layer of methane depends mainly on the gas liberation rate, the velocity of air flow and the dimensions of the working, as well as on other secondary factors To know beforehand whether in any given roadway a stable layer of methane will be formed, various indexes, or layering numbers, have been proposed. For example, use may be made of the expres-

sion

$$v \quad \sqrt[3]{4\,3\,\frac{Q}{D}}$$

where v = velocity of air flow, m/sec
Q = quantity of gas emitted into the airway, m^3/sec
D = width of the airway, metres

If the number obtained is more than two, layering will not occur. For example, let the velocity of air flow be 1 m/Sec. the liberation rate 0.2 m^3/sec, and the width of the roadway 2 metres, substituting these values in the equation above we obtain

$$1 \quad \sqrt[3]{4\,3\,\frac{0\,2}{2}} = 1.33 < 2\,0$$

It is therefore probable that gas will layer under the roof.

To eliminate the possibility of layering, it is recommended that.

(a) the air flow should be increased, which is the basic measure, and

(b) the air velocity should be increased by constricting the air stream, for example, by placing a hurdle across the roadway at the floor; placing obstacles at roof level is not satisfactory.

Gas Emission in Development Workings Driven in the Coal. This is composed of the gas emission from the face itself, from the broken coal and from the exposed coal surfaces throughout the length of the working During shotfiring, the liberation of the gas from the face increases abruptly (up to four times the gas emission of the seam), but then diminishes rapidly and for a long working drops to only a few per cent The main quantity of gas comes from the stationary exposed faces of the coal. Observations have shown that the quantity of gas given off by all the exposed coal surfaces is about 50% of the total gas emission for the length traversed in the previous month. for the previous month but one it is 25%, for the month previous to these two 12%, diminishing by a half for every month These data can be used to calculate the expected gas emission from development workings using the expression

$$q = \frac{4gA}{24\;60} \text{ m}^3/\text{min} \qquad (1\text{-}3)$$

where 4 = coefficient which takes account of the increased methane emission from development workings above the average methane emission of the seam through which the road is being driven;
g = methane emission of the seam being worked;
A = mean 24-hourly coal output from the development working, in tons, which is determined from the 24-hourly advance with a face 4 metres wide

The formula is valid for a working which has been driven for a period of time not less than five months. If the time for which the working has been driven is four, three, or two months or one month, the quantity of gas obtained must be multiplied by a correction factor equivalent to 0.93, 0 87, 0.75, or 0.50, respectively If two parallel roads are driven simultaneously and the width of the pillar between them is 30 metres or more, the total gas emission can be considered to be double that from one face.

Gassing up of Dead Ends. When the ventilation in a dead end is interrupted, the gas emitted begins to accumulate and its content can reach a dangerous limit; the dead end is said to fill with gas Unfortunately dead ends fill with gas fairly frequently and gas explosions occur in them. The main cause for dead ends filling with gas is the recirculation of air during the operation of an auxiliary fan, any other unsatisfactory condition of the ventilation, or a temporary stoppage of it. Recirculation in this sense is understood to mean the repeated circulation by the fan of the air coming from the dead end and blown back into it, even if the safety regulations are fulfilled, which limit the fan output to 70% of the air flow passing along the working where the fan is installed. One danger of gassing up is that when the dead end is being cleared of gas, the air coming out of it may contain a high concentration of methane exceeding the lower limit of explosibility (5%)

Underground observations on the filling of dead ends with gas have shown that·

(a) in the first few hours after the gas has begun to accumulate in the dead end, it accumulates in proportion to the time of exposure of the seam, that is, most of all in the face, gradually dying out towards the intake end of the working,

(b) for longer durations, exceeding three hours, the methane, being subject to the laws of diffusion, spreads more or less uniformly along the working;

(c) for even longer interruptions of the ventilation, almost undiluted gas can leave the dead end to join the airway at its entrance

Depending on the time for which the gas has been accumulating in a dead end, three methods of removing the gas are recommended

(1) for short durations (about three hours as maximum) and developments at least 50 m long, it is possible to feed air continuously into the gassy dead end;

(2) for longer durations it is necessary periodically to start and stop the auxiliary fan, depending upon the gas content in the return air coming out;

(3) finally, for long stoppages of the ventilation and more or less uniform emission of the gas along the working, it can be dangerous to deliver fresh air to the face; in this case the working must be

ventilated by short lengths, slowly building up the length of the ducting

Control of Gases Underground. Only recently, the only method of combating emission of gas underground was to dilute it with fresh air and take it out of the mine This control measure was effective for low yields of gas and not very high outputs, but it failed to handle the vast quantities of gas met with in modern large and deep mines The air requirements became so high that ventilation costs rapidly increased because of the higher power of the fans and the need to drive large underground airways to satisfy the permitted air velocities (not more than 8 m/sec in main haulage roads, and not more than 4 m/sec in the face)

Therefore it became necessary to reduce the total quantity of air delivered to the mine by removing part of the gas to the surface, with the mine workings being by-passed, by eliminating the abrupt increases of gas emission during the work of cutter-loaders and by redistributing the gas emission in time. The measures aimed at solving this problem have the general name of control of gases and at the moment are extremely important.

To reduce the total gas emission, a relatively new technique, called *methane drainage*, is used at present for removing methane from coal seams This method will be discussed separately

To lower the fluctuations of gas emission in the face, use is made of the measures mentioned above in considering gas emissions caused by cutter-loaders

Appropriate roof control can help by keeping the gas emission from neighbouring seams mainly in the worked-out area and not in the face, thus helping the ventilation

Reduction of the methane content in the return air flow of the mine, of the district and of the face A high gas content in the general return air of the mine can be caused by very high gas emissions not forecast in the planning, or inadequate air flow from the fan Methods of reducing the gas content include firedamp drainage; increase of the air flow by increasing the fan speed or the angle of attack of the fan blades, changing the fan for a more powerful one, reduction of short circuits in the air flow, even if they reach 30-40% does not greatly affect the quantity of air passing through the face

A high gas content in the district return air can result from similar causes, methods of reduction of the methane content include firedamp drainage, equalization of the fluctuations of gas emissions in the face, in order to increase the air flow into the district, leakages of air on the way in must be reduced, the air distribution between districts should be changed or a booster fan installed

A high gas content in a return end of a face can be caused by a large gas emission in the face or by an inadequate air flow reaching

it. Methods of reducing the gas content include. firedamp drainage; reduction of the fluctuations of gas emission in the face; reduction of the face length; subdivision of a face which is one horizon in height into sub-horizons; changing the ventilation layout, preventing the gas passing from the worked-out area into the face; in order to increase the air flow into the face, air losses should be reduced within the district and the air redistributed between the neighbouring faces, to increase the quantity of air flow in the return airway, with mining by sub-horizons, some air can be borrowed from the intake airway of the sub-horizon above, or an additional quantity of air can be supplied from the haulage road to the return airway along the stenton which connects these roads, using an auxiliary fan and ducting

Finally a combination of these methods is also possible. In particularly difficult conditions it is not impossible that the speed of advance of the face may have to be reduced or the main mine fan may have to be changed for a more powerful one. Such changes must necessarily be fully planned beforehand.

Dangerous and permissible contents of methane in the mine air. As indicated above, a mixture of methane and air becomes explosive at 5-6% of methane. It is quite natural that Soviet Safety Regulations provide for considerably lower methane contents in the air, namely:

In the general return air of the mine	not above 0 75%
In the general return air of the district, of the face and of development roads .	not above 1 0%
In the air intake into the face of development . . .	not more than 0 5%
The use of explosives is allowed at a content of	not more than 1 0%
Local (in individual places) accumulations in the faces, in the development roads and elsewhere . . .	not more than 2 0%
Workmen must leave the face when the content exceeds	2 0%

1-6. GAS BLOWERS

Blowers are violent jets of gas released from cracks, usually accompanied by hissing, and if the gas comes from beneath water, by strong bubbling sounds, audible over considerable distances in the silence of the mine Frequently blowers first throw out water and then gas

Blowers are most often found in places which have been folded or faulted, but are extremely variable both in duration and in

output. Some of them last only a few days or months, others last years or tens of years The output from a blower varies from several m^3 up to some thousands or tens of thousands of m^3 per day, but, as a rule, its output is a maximum at first and gradually falls until it is exhausted

The flow from a blower depends on the dimensions of the opening through which the gas passes, as well as on the pressure. The pressure at the beginning of the release of gas from blowers can measure several kg/cm^2.

Since the gas from blowers is usually nearly pure methane (80-90% or more), powerful "permanent' blowers are sometimes used for scientific purposes or for lighting or heating at the surface of the mine.

A well-known large blower was met with in the sinking of a pit at the British coal mine of Garwood. The gas was led to the surface and burned for nine years in a flare so high that the flame could be seen 15 kilometres away

Blowers in the USSR have been recorded which have given hundreds of thousands or even millions of m^3 of gas, some of them were active for years, giving at first up to 20,000 m^3 of 100% methane per day, dropping to 8,000 and 8,500 m^3 after about 500 days, in this time the blowers gave over 6 million m^3 of methane

Since blowers are met with unexpectedly in the driving of roadways, the meeting of a large blower is extremely dangerous, since not only the individual drivage but also whole districts of the mine can be filled with mixtures of gas and air which may be suffocating as well as explosive

Blowers are usually found in certain geological structures but they are by no means confined to them. Under Soviet Safety Regulations, mines which work seams subject to blowers should make use of advance exploratory boreholes These boreholes are drilled through a so-called blowout preventer, a device which enables the borehole to be closed if it meets a blower. The blowout preventer is solidly cemented into the collar of the hole. If greater safety is required during the drilling, the drilling rig can be remote controlled and men do not then need to be at the face of the drivage

When blowers are met they can be dealt with in one of the following ways.

(a) they can be closed in, that is, the crack from which they issue is blocked up,

(b) the gas can be drawn off through a pipe by the shortest way direct to the surface; in this case, work in the return airway is forbidden, except for what is necessary for ventilation, repairs and the supply of materials,

(c) if the blower is not very large, it is allowed to die out naturally, but the ventilation is simultaneously increased in the district.

1-7. SUDDEN OUTBURSTS OF COAL AND GAS

Sudden outbursts of coal and gas are included among the dynamic phenomena of coal mines, those with a violent effect. These include rock bursts and similar incidents caused by the pressure of the overlying rock; inrushes of water, gas or mud; sudden outbursts of coal and gas resulting from the joint action of rock pressure and gas pressure.

Sudden outbursts have the most powerful effects and are most complicated in their origin. Their origin and development depend not only on the rock pressure and gas pressure but also on the structure of the coal seam, the properties of the coal, the angle of dip and the thickness of the seam.

When development roads and coal faces are driven forward, the stressed condition of the seam changes. Next to the face the coal cracks, partly breaking up, becoming more friable and losing most of the gas contained in it. In this face area the stress in the seam is not large

At some distance from the coal face (about 1-3 metres) the coal becomes more dense under the effect of the rock pressure and capable of carrying a considerable load In this region of so-called abutment pressure, the gas permeability of the coal is reduced, forming a gas barrier which prevents the emission of gas into the face. At some distance from the exposed surface of the seam the coal and the gas contained within it are under a considerable pressure.

As the coal face advances, the wave of pressure advances ahead of it together with the gas barrier.

If the structure and strength of the coal seam and adjacent rock along the strike do not change, this process continues uniformly and the gas flows with a more or less constant velocity out of the coal face where the coal is cracked

If, as often happens in coal mines, the structure and strength of the coal seam and adjacent rock are variable, the roof on top of the workings may sink not smoothly but jerkily, causing additional dynamic loads on the coal seam. These loads are particularly heavy if, above the seam, the rocks are strong, hanging over the worked-out area or if the coal seam has a complicated structure with beds of strong coal and partings of deformed weak coal or carbonaceous argillite.

When additional dynamic loads fall on the coal seam or sudden fractures of hard inclusions occur within it, part of the coal seam breaks up near the zone of abutment pressure and gas barrier the kinetic energy obtained by the coal from the rock and its accumulated potential energy are expended on crushing the coal and throwing it out with its accumulated gas; during the crushing, the surface

area of the coal increases enormously, forming instantaneously a very large number of channels for the release of gas. The gas is violently emitted from the crushed coal and during its expansion and movement pulls the coal out with it If the seam is inclined and the working is driven to the rise, the weight of the coal increases both the speed and the distance of ejection As a result, the working sometimes fills with coal for tens of metres and with gas for hundreds of metres

The hazard of sudden outbursts in coal seams increases with diminishing strength of coal and with increasing thickness of the bands of weak, deformed coal, with increasing thickness of the seam and with increasing angle of dip Where geological disturbances have taken place, sudden outbursts are more frequent than in areas where the seam lies quietly, in development roadways they are more frequent than in production faces; but the intensity of sudden outbursts in production faces is greater than in developments in the coal, the greatest intensity of sudden outbursts usually occurs where a seam is entered by a crosscut

The overwhelming majority of sudden outbursts take place from above downwards, but in some rare cases they have occurred from below upwards Some hundreds or even tens of thousands of m^3 of methane participate in sudden outbursts Typical emissions per ton of coal thrown out vary from 10-60 m^3 and they generally stop rapidly, but sometimes the gas emission continues for a long time after the outburst and apparently takes place from blowers exposed by the outburst

Voids of the most varied forms result from sudden outbursts. Often the void is pear-shaped with the long axis up the dip of the seam (Fig 1-9)

The ejected coal is in part very finely ground, sometimes covered with a layer of fine dust Sometimes the coal is squeezed out into the working in one large block, having a volume of several m^3

Theory of Sudden Outbursts. From the outset the origin of sudden outbursts has been explained by various theories Theories are needed, since they give the causes of sudden outbursts and if these are known it may be possible to suggest how to prevent them.

The main theories of sudden outbursts are briefly considered below. They can be divided into theories based on gas and those based on rock pressure According to one of the earliest theories the cause of sudden outbursts is the pressure of the free gas contained in the coal, the other theory states that the coal is thrown out by the pressure of desorption of the gas from the coal when the load is taken off the coal (near its exposed surface) The supporters of the theory based on rock pressure as the cause of sudden outbursts believe that the main factor affecting the onset of a sudden outburst

is the weight of the overlying rock, and the stressed condition of the seam and its roof resulting from this weight; others consider that the stressed condition of the rock is a result of tectonic processes, which may be ancient or taking place at the present time, and that the forces acting on the coal seam and surrounding rocks are directed along the bedding plane of the seams; this point of view is supported by the fact that sudden outbursts most often take place near disturbances in the bedding. A delay in lowering or caving of the roof

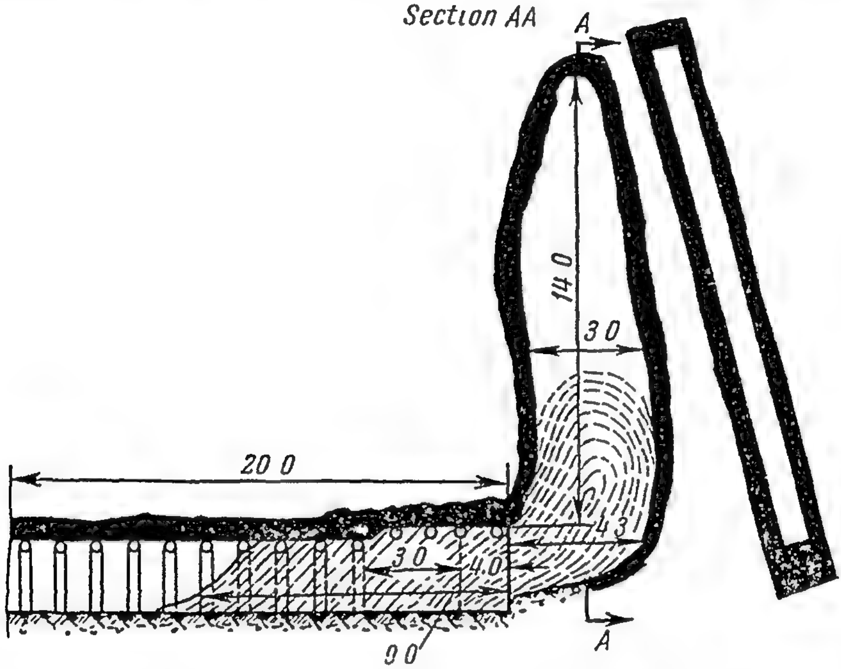

Fig. 1-9 Diagram of the hollow formed by a sudden outburst at a development

over the waste is indicated as a possible cause of sudden outbursts; with a sudden fracture of the overhanging roof, the coal is crushed, and a sudden outburst results Some importance should be also attached to the weight of the coal and to the properties of the coal seam

At present it is difficult to give preference to any one of these theories because none of them contradict the facts of underground work Every one of the eauses of sudden outbursts is a possible one, under certain conditions each of them can lead to a sudden outburst

Much effort has been spent on establishing the distinguishing properties of seams subject to sudden outbursts. If it can be established in good time that a particular seam may be subject to sudden outbursts, this knowledge may be very valuable from the safety

point of view Unfortunately it has to be stated that there is no generally accepted indication that a seam may be subject to sudden outbursts It is considered that the initial rate of gas liberation from hazardous seams is high, that their volatile content is low, and that their gas pressure is high, seams subject to sudden outbursts generally have a complicated structure.

In addition to outbursts of coal, some mines have suffered outbursts of rock Thus at one mine in the Donets Basin at a depth of 915 metres in 1962 and 1963, during the driving of two crosscuts, 12 outbursts of rock took place without any considerable emission of gas, the maximum quantity of rock thrown out was 2,100 tons, forming voids measuring 8 5 × 9.5 metres. All the outbursts took place in a district with a thick sandstone in the roof, where the dip changed from 18° to 30-40°, and a probable cause of the outburst was the unloading of the stress in the sandstone, originated by the folding

To ensure safe working conditions it is extremely important to know the preliminary signs of sudden outbursts. They can be divided into (a) forerunners which show that a sudden outburst can happen, and (b) the immediate warnings that precede an outburst The forerunners include. change of thickness, structure or strength of the coal (generally a reduction), increased gas emission; cracking sounds from the coal mass, gripping of the drill during the drilling of holes, and other signs Immediate warnings include bumps or shocks from the coal, sounds like booms or rumbles; small pieces of coal jumping out, squeezing out of the coal seam towards the face.

Recently various instruments have been used for early recognition of the possibility of a sudden outburst, namely. geophones, mine tape recorders combined with them, and seismometers

Prevention of Sudden Outbursts. Here we shall consider the measures taken when hazardous seams are first entered, those taken in development roads, and finally the practice in production faces.

When seams subject to sudden outbursts are first entered, even if the seam is only 0 15-0 2 metre thick, violent outbursts are likely to occur, therefore the most serious attention should be paid to this responsible task Entry takes place in two stages The first stage consists in the drilling, at 10 m from the seam, of two exploratory holes at right angles to the bedding plane, and in the intersecting of the seam for its full thickness, at this stage the gas pressure is measured in the seam and then reduced to 10 atmospheres At 6 metres before the hazardous seam, the cross section of the development (in a steep seam) must be reduced to 5 m²

The second stage consists of the actual entry. The instructions for safe mining practice in seams subject to sudden outbursts of coal and gas recommend the following methods of entry

(a) preliminary extraction of another seam immediately above the hazardous seam; after the production face of the upper seam has advanced twice the thickness of the intervening rock, beyond the point of entry into the hazardous seam, entry into the latter is permitted with the use of shock blasting, also called inducer blasting (see below);

(b) the use of firedamp drainage holes; these must pass through all the rock and the whole coal seam; the number required is determined experimentally; after the gas pressure has been reduced to 10 atmospheres, entry into the seam is by shock blasting;

(c) preliminary water infusion into the coal seam; boreholes 40-45 mm in diameter are drilled into the seam and water is forced into them under a pressure of 30-50 atmospheres; when the gas pressure is reduced to 10 atmospheres, the seam is entered by shock blasting;

(d) by using a hydraulic monitor to enlarge a hole 250 mm in diameter drilled in the seam; the water washes out the coal, forming a hollow in the seam, before firing the shots, the hollow is filled with water;

(e) by the use of a steel frame, this method is used for entering seams in which the pressure reaches 30-40 atmospheres and cannot be reduced to the required 10 atmospheres; the steel frame is built around the roadway by drilling one or two rows of boreholes 65-70 mm in diameter, so arranged as to intersect the coal seam and to be sunk in the floor (or the roof) to a depth of 0.4-0 7 m; steel pipes with a diameter of not less than 50 mm are pushed into these holes and under their projecting ends masonry or steel arches are built, the rock plug so formed being removed by shock blasting

In seams subject to sudden outbursts, development roads are driven by the following methods:

(a) By drilling advance holes, originally, holes of a small diameter (37-42 mm) used to be drilled, on the assumption that enough gas would be drained off through these holes to lower the gas pressure and to prevent sudden outbursts; experience of drilling these holes for many years has shown that they are not effective, and present universal practice in the USSR is to drill advance holes 250-300 mm in diameter; the results of drilling these large holes are: first the face of the coal seam is drained of methane; secondly the stress in the coal is reduced. In gently dipping seams, depending on the size of the roadway and the thickness of the seam, from two to five boreholes, 10-20 m long, are drilled at an angle from 7 to 21° to the horizontal; in steep seams the number of boreholes varies from 1 to 5, their length being from 12 to 21.5 m and their inclination 13-35°; the overlap between one set of holes and next must be strictly maintained at not less than 5 m. The distance between the holes

must be uch that their zones of effectiveness within the seam overlap generally the distance varies between 0 5 and 2 0 m.

(b) By the use of shock blasting (also called inducer shotfiring which consists in shotfiring by overcharged holes in the absenc of men, artificially shocking the coal mass and the enclosing roc and creating the conditions for a sudden outburst The first shoc blasting was used in France in 1890, after which it was widel used not only in France, but also in Belgium. Britain and Germany shock blasting is now the main method of preventing sudden out bursts in developments; in the USSR it was first used in 1933

During shock blasting in coal in all roadways into which methan can be emitted after blasting, the electric power must be switche off, the men must leave all return airways and other nearby area and go to a safe place not less than 1000 m away, or if this distanc cannot be achieved, to the surface; during shock blasting in coa the shotfirer must be at least 600 m from the face in the intak airway (for blasting in rock the distance may be reduced to 200 m Before shock blasting it is necessary to distribute water at the fac in polythene bags at a rate of 5 litres per m^2 of roadway section the dust in the road must be bound with a wetting agent for a distanc of 20 m from the face, primary stone-dust barriers must be bui (see below) at a distance of 15-30 m from the face, any detonato can be used, but if short-delay detonators are used their total dela time must not exceed 120 milliseconds

The schedule of shock blasting varies with blasting condition for the Donets Basin coalfield the following are recommende 2-6 boreholes as a minimum per m^2 of the area of the face fire length of holes 1.5-3 m, total charge per hole 0.4-1.2 kg, total weigh of the charge in all the holes 2.4-18 kg

Some disadvantages of shock blasting include the possible ign tion of methane or coal dust, possible delayed outbursts during th working of the coal, the roof can be shattered in weak ground, th number and intensity of the outbursts increase, the cost of drivin the roads is high because of the inevitable extra work

(c) By the preliminary mining of protective seams, this meth is the most reliable, and if there is a seam close enough to the o subject to sudden outbursts, it is used for the opening, developme and working of the seam subject to sudden outbursts Both theo and practical experience have shown that in the protective minin of a neighbouring seam, whether below or above the hazardo seam, changes occur in the stress in the seam and neighbourin beds and there is a reduction of the gas pressure in the coal an a partial loss of the gas from the hazardous seam. These chang reduce or completely eliminate the danger of sudden outburst This action is described as protective, and a seam thus mined ahea

of a hazardous seam is described as a *protective seam*. The district of the hazardous seam which is protected by the other seam is described as a *protected zone*. The protective action of advanced working of neighbouring seams is now considered proven for a distance of 80 m at right angles to the bedding planes between steep seams, and between gently sloping seams, for as much as 170 m with a protective seam beneath.

Current Soviet Safety Regulations state that "seams which are threatened by sudden outbursts may be worked *only after the preliminary extraction of a protective seam*". The productive face work in the protective seam must be ahead of the hazardous seam by not less than twice the spacing between seams measured normal to the bedding plane.

If the mining of the hazardous seam is delayed for any reason, the rock between the seams begins to consolidate, but mining experience has shown that this has hardly any effect on the protection given by the preliminary working.

During the extraction of protective seams we must remember that part of the dangerous seam remains unprotected, that is.

(a) with a protective seam above, the upper part of the horizon for a distance of 0 55 H, in which H is the perpendicular distance between the hazardous seam and the protective seam (when the protective seam is not mined in the upper horizon), and

(b) with a protective seam beneath, the lower part of the horizon for a distance of 0.55 H, when the protective seam is not mined in the lower horizon, and is at a perpendicular distance of mor than 10 m.

The efficiency of mining by protective seams is indicated by the fact that in the Donets Basin from 1948 to 1959 there were no sudden outbursts in roadways driven in protected areas.

The last method used for driving roadways in seams subject to sudden outbursts consists in the use of special supports which prevent coal in steep seams from spilling out; this special support consists of three to five timber props of 120-150 mm diameter or steel tubes of 50-75 mm diameter driven into the holes drilled at 8-10° inclination in the upper part of the face of the roadway; the length of the props is 2.5-3 m and of the tubes 7-8 m; under the projecting ends of the props or tubes, bars are placed.

The following measures should be taken when producing coal in seams subject to sudden outbursts:

(a) Obligatory preliminary mining of protective seams where these exist.

(b) The face is kept straight (to reduce the number of ribs or re-entrant corners in which the stressed state of the seam and adja-

cent rock is the greatest); the haulage road and the face are in one straight line.

(c) The face is parallel to the strike, and worked to the dip so that the weight of the coal is used to reduce the likelihood of sudden outbursts.

(d) Faces should be the length of the horizon and not divided into sub-horizons so that along the line between the coal face and the worked-out area, the stress in the enclosing rock and the seam being worked shall be reduced to the minimum.

(e) Faces should be extracted (retreating) from the boundary towards the crosscut because the driving of the developments releases part of the free gas, and the stress in the seam and the enclosing rock diminishes; the development roads should then be driven in the stone.

(f) Full caving of the roof should be used so as to prevent the roof from hanging over the waste.

New proposals for the prevention of outbursts include:

(a) Chambering shotholes in the coal mass; this consists in drilling holes every 7-10 m for the full depth of the horizon ahead of the face, to a diameter of 120 mm, charged with 63% dynamite at a rate of 2.2-3 kg per one metre of hole; the boreholes are filled with water, sealed, and detonated by detonating fuse.

(b) Hydraulically squeezing out the coal in the seam: throughout the length of the face, holes 5 m long are drilled every 4 m and water is injected into them at a pressure of 250-350 atmospheres.

(c) Preliminary firedamp drainage of hazardous seams.

1-8. METHODS OF TESTING THE MINE AIR FOR METHANE CONTENT

1. Methods of Testing. The mine air is tested for methane by the following three methods:

(a) by taking air samples in the mine and chemically analysing them in laboratories at the surface;

(b) by directly testing underground by means of flame safety lamps, either ordinary or with a device to make the gas estimate accurate, or by the use of special indicator lamps;

(c) by the use of special instruments: gas analysers, and so-called indicators or detectors which indicate methane or determine its percentage content in the air by volume.

2. Sampling of the Mine Atmosphere. The simplest method of sampling mine air consists in the use of a plugged flask, filled with water at the surface and emptied at the sampling point, the water flows out and its place in the flask is taken by the mine air. After the water has flowed out the flask is stoppered with a cork

or a rubber bung; if a cork is used the cork and the neck of the flask are greased with some airtight composition. This method is unreliable: air can leak in while the sample is being stored; there is also a possibility that the flask is dirty, in which case the analysis will be inaccurate.

Air samples should therefore be taken in special glass tubes, such as pipetters or burettes, of 150-300 cm^3 capacity, sealed at both ends to small capillary tubes with an internal diameter of about 1 mm, stoppered with carefully ground glass cocks (Fig. 1-10) or with rubber tubes closed by clamps. These containers are numbered, filled with distilled water and taken underground, several at a time, in special closed boxes. At the sampling point, the upper valve is first opened, and then the lower; the water gradually flows out and the air to be tested flows in.

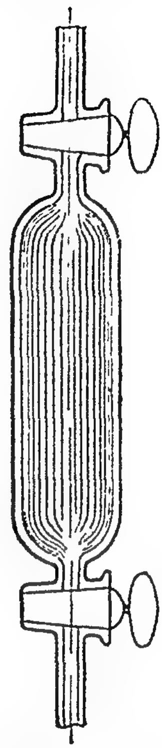

Fig. 1-10. Air-sampling tube for mine air

If it is desired to take an average sample throughout the air flow, then while the water is slowly flowing out, the tube is moved throughout the cross section of the air flow, covering as many points as possible. Do not shake the tube when taking the sample so as to prevent the carbon dioxide gas dissolving in the water.

Experience has shown that air samples can be stored in these tubes for several months without change, provided that the cocks are well ground in and greased. The best grease for this purpose is heavy mineral oil, preferably with a small quantity of lead soap.

However carefully the water is released from the burette, the inner face will carry a film of water, and because carbon dioxide is easily absorbed by water the sample may be faulty.

Therefore, when the carbon dioxide content is required as well as the methane content, it is better to take the sample in a dry pipette, forcing the air out of it with the aid of a small rubber bulb, fitted with a blowing valve and a suction valve. If the amount of air drawn through the tube is more than 50 times the capacity of the burette, one can be confident that the original air has been expelled by the mine air. For taking samples by this method use is made of tubes not with a capillary end but with wide fittings. These tubes can also be used for taking samples with water.

Recently *vacuum burettes* have been used, particularly for taking samples from one point or for sampling a uniform mixture of gas and air; the air is pumped out of these burettes, their ends being

drawn out to capillaries and sealed. When the sample is taken, one capillary is broken off, the tube immediately fills with air, and the end of the tube is closed with a rubber cap.

When samples of air must be taken in a particular place several times at uniform intervals in a day or a shift, automatic samplers can be used. One of these has been developed by VUGI, the All-Union Coal Research Institute

The operating principle of the automatic sampler designed by VUGI is clear from the diagram (Fig. 1-11). There are two metal containers *1* and *2*, the former being a burette for taking the samples, placed above *2*, and provided with two thin tubes *3* and *3′* (2-3 mm diameter) with valves *4* and *4′*. If burette *1* is filled with water and the valve is opened, water begins to flow out of it into container *2* through tube *3*; through the tube *3′* air is sucked, filling burette *1* in proportion to the amount of water which flows out; this will continue until the water level in container *2* reaches the level *pq* at which the entry to the aspirator tube *3′* is fixed; this closes the entry to the aspirator tube and no more air can enter burette *1*; this completes the sampling. The depths and diameters of containers *1* and *2* are so designed that the water level in container *2* rises to the lower level of aspirator tube *3′* and closes it just as burette *1* is almost empty.

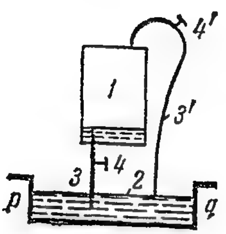

Fig 1-11 Automatic air-sampler

Portable Gas Analysers. The methane content is usually measured in the laboratory by the method of burning the gas, using the instruments of B E Dolgov, Brokman and others

The accuracy possible in methane determination by chemical analysis in these instruments is about 0.05% if the mixture is non-explosive and if it does not have to be diluted during analysis; otherwise, the accuracy is less.

Chemical analysis of air samples by these instruments is the most reliable and accurate method of determining the methane content of the mine air. Some modern mines take tens of air samples every day inderground, which are analysed the following shift in the laboratory so that the results are available within one shift. Frequently in practice, however, the gas content in the faces and elsewhere must be known immediately. With this in view, portable underground gas analysers have been proposed, of which we can mention the MakNII, type MC-2. The instrument functions in the same way as stationary laboratory gas analysers, by the burning of methane in the air sample and measuring the reduction in volume in the combustion chamber after the cooling of the carbon dioxide and nitrogen so formed.

The MakNII instrument (Fig. 1-12) consists of the following parts: a pump (rubber bulb) for taking the sample and blowing through the combustion chamber *1*; a diaphragm manometer *2*; a combustion chamber *4*, filled with distilled water up to the level of its neck, and a cock *3*. In the upper part of the combustion chamber *4* a thin platinum spiral is fixed which is heated by a 2.5 volt alkaline accumulator *5*. The pump, the chamber, and the manometer are mounted in a small aluminium box; the accumulator is not in the box.

The sequence of work with the gas analyser is as follows.

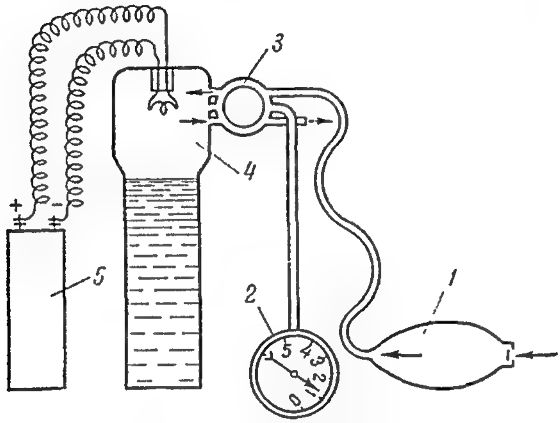

Fig. 1-12 Diagram of the MakNII portable methane analyser

Before going underground for firedamp measurements the condition of the instrument is checked as well as the voltage of the accumulator, which should be equal to or above 2.5 volts.

At the measurement point the valve *3* is opened and the chamber is blown through by ten squeezes of the bulb. The chamber is thus filled with the air sample. The valve is then closed, the current is switched on and the chamber is held in this condition for two minutes; the chamber is then allowed to cool with the gases in it for not less than two minutes. By the cock *3*, the chamber is then connected with the manometer; the pointer travelling across the manometer scale indicates the methane content in the sample, the cock *3* is again turned to disconnect the manometer from the chamber and to connect the chamber with the outside air; the manometer pointer then returns to zero.

The instrument is suitable and gives accurate results for methane contents from 0.25 up to 4% provided that the working parts of the instrument are in perfect condition and that the instrument is properly used.

The time required to make one measurement is about ten minutes. The instrument is compact, and apart from the accumulator, weighs about 2 kg.

One practical and relatively important disadvantage of the instrument is that its accuracy depends on the accumulator voltage and on the temperature conditions (the temperature differences between the gas mixture before and after burning and between the mixture and the instrument itself). Not less than 2.3-2.5 volts should be available from the accumulator which serves the instrument. The accumulator requires recharging after 250 tests.

The temperature differences mentioned above (between the instrument and the mixture being tested as well as between the samples before and after burning) significantly affect the accuracy of the instrument readings. Thus, for example, a difference of 1°C, at a mine temperature of 10°C, gives an error of 1.17%.

It is also important that the volume of air in the sample drawn into the combustion chamber should be precisely that for which the manometer is graduated and therefore the experimenter must be careful that the water is always at the correct level.

The electrical gas analyser for methane functions on the principle of electrical measurement of the difference in the thermal conductivities of pure air and a mixture of methane with air. The accuracy of measurement of the methane content by such a gas analyser does not exceed ±0.3%.

Mine Interferometers. These instruments are designed to determine the methane, carbon dioxide and oxygen contents in the mine atmosphere. The operation of the instruments is based on the principle of shifting of an interference pattern due to differing refractive indices of air and a mixture of methane and/or carbon dioxide with air, through which one of the coherent rays passes

Figure 1-13 is a general view and an optical diagram of an interferometer of the ШИ-5 type which is a portable instrument for direct determination of the methane and carbon dioxide in the workings.

Outside the instrument casing are. a connecting pipe *1* for drawing air into the instrument; a connecting pipe *2* which is inserted into the rubber bulb *7*, an eyepiece *3* with a protective cap, a screw cap *4* which serves to plug the inlet branch pipe connected with the filter filled with a chemical absorbent for carbon dioxide, an adjusting handwheel *5* fitted with a cap for fixing up the spectrum in the zero position, a window *6* for directing the light into the instrument from the miner's head lamp.

The optical system of the instrument is as follows the light from a small electric lamp (MH-3) *1* passes through a condenser lens *2* to project a pencil of parallel rays on to a right-angled, totally

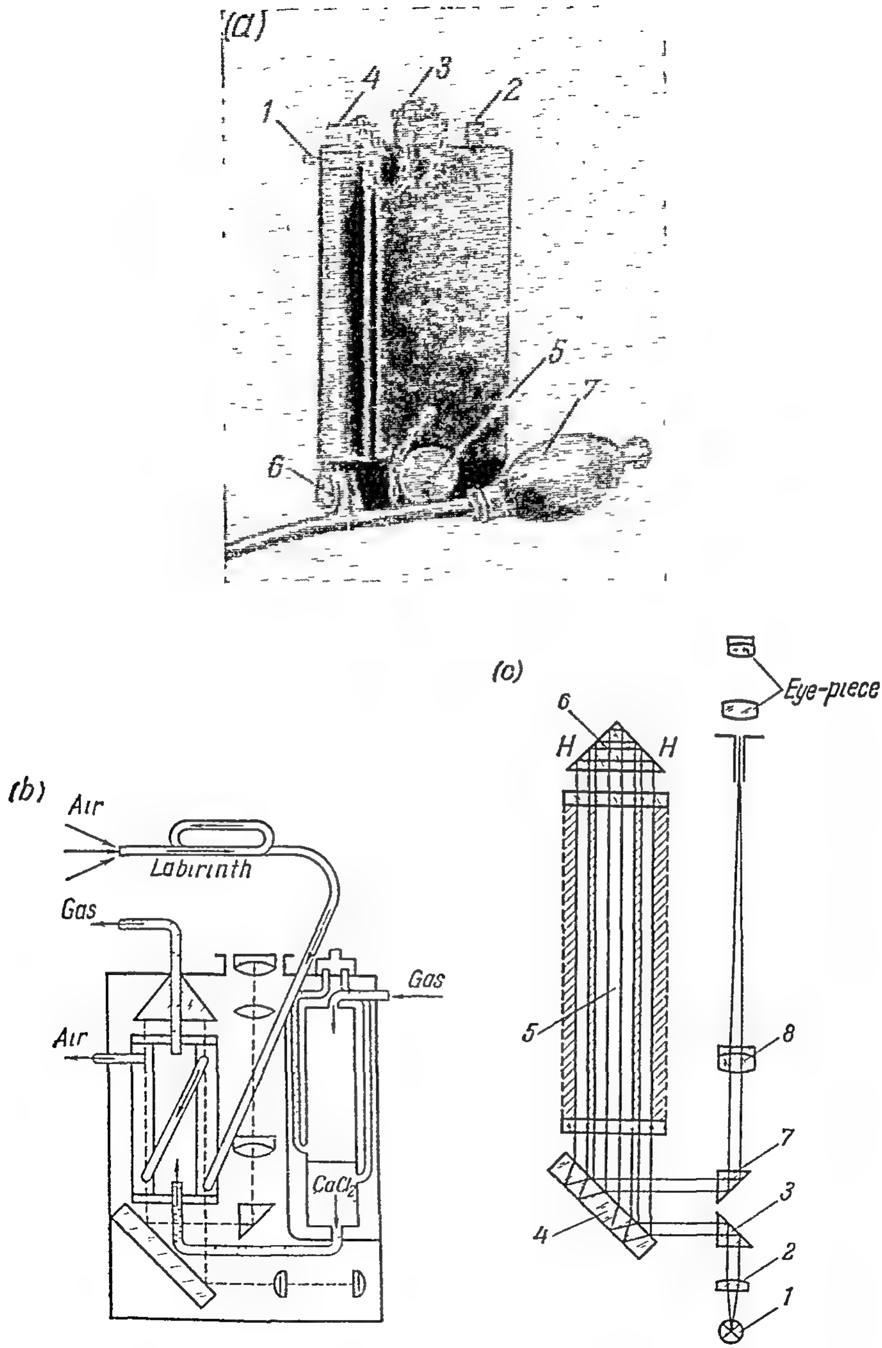

Fig. 1-13. Interferometer ШИ-5

reflecting prism *3*. From the prism the light is reflected on to the plane-parallel plate *4* with a silvered lower surface. One part of the rays is reflected from the outer surface, the other part from the internal surface of the plate *4*, and both pencils of light after their second turn through 90° are directed through the two compartments of the gas chamber *5* to the second totally reflecting prism *6*; here they are reflected twice and then returned to the plate *4*, the first pencil of light passing through the outer compartment of the gas chamber and the second through the inner compartment, as before. After passing in the reverse direction through the gas chamber, both pencils of light are again reflected, the first from the internal (mirror) surface of the plate *4*, and the second from the outer surface; they are then directed towards the third totally reflecting prism *7*; after turning through 90° the pencils of light pass through the objective of the viewing tube *8* and are collected at the focal plane of the tube *8*

If in the middle part of the gas chamber we place the gas mixture to be investigated, having optical properties which are different from the control gas mixture (for example, pure air), the second pencil of light will pass through the medium with a different refractive index, the path differences change, and the interference fringes visible in the eyepiece change.

So as to restore the original interference pattern, the path difference of the light must be compensated by rotating the prism *3* by an angle on its axis

The gas system of the instrument consists of two lines: one for the gas and the other for air The gas line is shown in Fig 1-13*b* by the dotted line.

From 350 to 400 measurements can be made without replacement of filters The range of measurement is 0.25 to 6 per cent and the accuracy is ±0 25 per cent

The ШИ-6 interferometer can also be used for determining oxygen within the range of 9 to 21 per cent, the error of measurement being ±0 3 per cent.

The ШИ-3/100 (or ШИ-7) interferometer is designed for determining high (up to 100 per cent) concentrations of methane and carbon dioxide with an accuracy of 2.5 per cent.

The *stationary continuous gas analyser* type ИМ-2 gives local and remote signals of high methane contents underground If the methane content exceeds a safe limit, the instrument switches off the power supply to the district through a control device

The main element of the ИМ-2 is a detector which feeds current continuously into the system, proportionally to the methane content of the air. The value of the current is shown by a pointer on a scale calibrated in per cent of methane. In parallel with the pointer

is a magnetic amplifier, increasing the current up to the value required for operating the primary relays. The detector works on the principle of electrical measurement of the heat from the burning of the methane drawn into the combustion chamber with the mine air

The ИМ-2 automatic gas analyser consists of three main parts (Fig. 1-14): (a) a methane detector, type ДМ-3, (b) a signal and control apparatus for methane, type АСМ-1; and (c) a signal board, type ТСМ-1.

The methane detector is installed in the return airway from the face or in another place underground where it is desirable to monitor

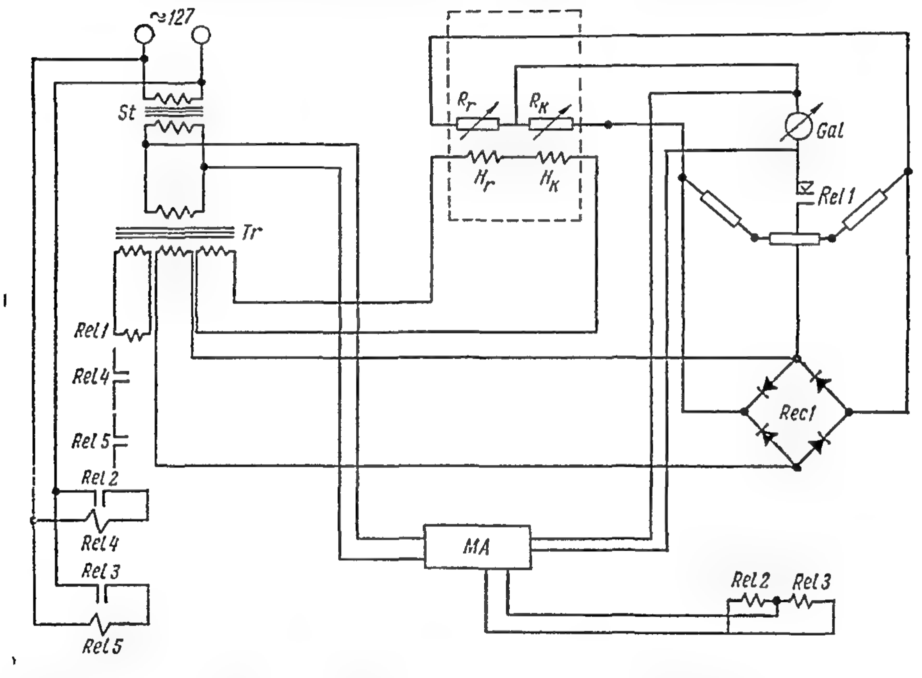

Fig. 1-14 Circuit diagram of the continuous methane analyser· *Rel 4*, *Rel 5*—operating relays, *Rel 2*, *Rel 3*—primary relays, *Rel 1*—time relay, *1 Rec*—selenium rectifier, *MA*—magnetic amplifier, *Tr*—step-down transformer, *St*—ferro-resonant stabilizer, *G*—galvanometer, H_r and H_k—thermistors, R_r and R_k—resistance thermometers

the air stream continuously. The signal and control apparatus for methane is installed in the haulage road, the signal board being in the control room.

The methane detector is a dismountable aluminium combustion chamber, within which the thermometers and their heating elements are fixed. The detector is 240 mm high, 250 mm wide, and weighs 7 kg without the casing.

The methane signalling and control apparatus consists of automatic elements, including an electric supply for the detector and an interlocking switch housed in a welded metal box. In the upper part are the methane indicator and alarm siren. The apparatus is 850 mm wide and 800 mm high.

The signal board is furnished with a warning siren and is contained in a metal box with the secondary relays for signalling and the signal lamps inside it. One board can serve five detectors.

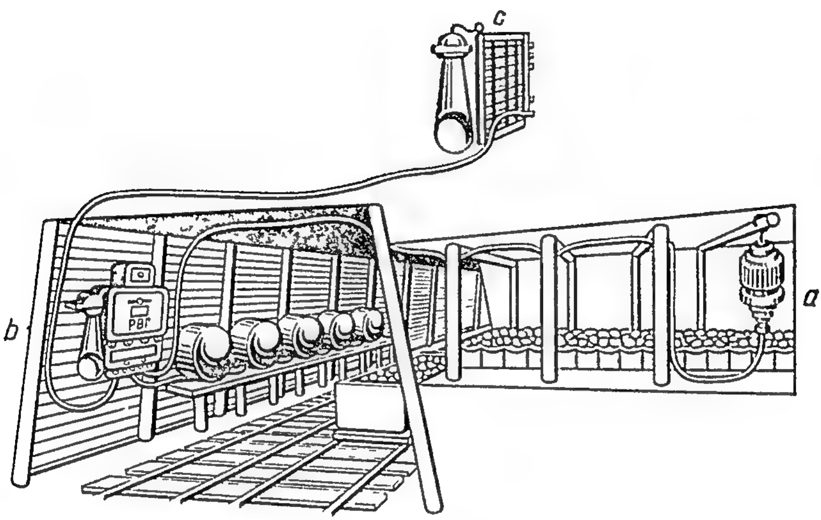

Fig 1-15 Automatic methane detector, type ИМ-2

On the lid of the board are 20 lamps with tinted lenses (four lamps for every detector) indicating the operation of the instrument. The accuracy of measurement and of response of the whole unit is 0 2% methane.

The stationary automatic methane detector type ИМ-3 differs from the ИМ-2 instrument in that it does not require a special cable and consumes the high-frequency current from telephone cables.

The *stationary thermocatalytic methane relay* type АМТ-2 is used for continuous automatic monitoring of the methane content in exhaust air. The instrument gives alarm signalling and shuts off electrical power to face equipment when the methane content in the air exceeds a safe limit The operation of the methane relay is based on the flameless burning of methane on the surface of a carrier catalyst at a burning temperature of 360°C.

In mines in Britain and other countries, Type "M" Spiralarms are widely used Subjected to every condition of regular service, they have been absolutely satisfactory in every way.

Design of the Spiralarm Methane Indicator (Figs 1-15 and 1-16). A circular wick burner *4* draws oil from the fuel container *11* (Fig. 1-15). The filling screw *5* and an air chamber in the fuel container also act as compensators to maintain flame height constant at all oil levels. The contact tube *10* contains two contact strips, each having a contact point. The tube has a slot at the level of contact pin *9*, which closes the circuit under the conditions described below. The pillar *3* carries a pointer *2* which is a flame-height setting gauge and also the simple bimetallic spiral *7*, which gives the indicator its name "Spiralarm". This highly sensitive spiral is made of bimetallic strip specially chosen for its remarkable physical properties and absolutely consistent response to varying temperature. Rising temperature due to increase in flame size causes the spiral to unwind and carry its pin *9* towards the contact tube and the slot in the tube. The falling flame temperature reverses the action Under the fuel container is the battery compartment *12*, housing a small long-life dry battery and a small electric lamp. A red glass window surrounds the aperture between the battery compartment and the base There is a switch *6* for testing purposes

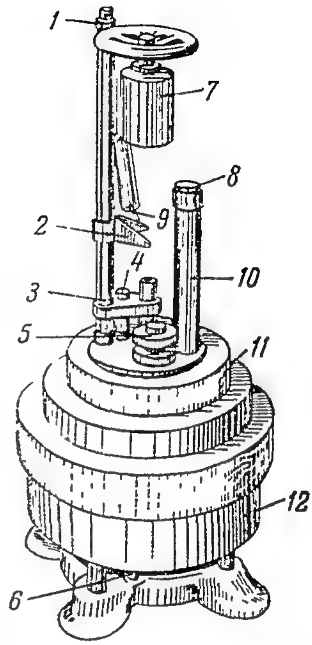

Fig. 1-16 Spiralarm methane detector type "M"

1—nut, *2*—height indicator for flame, *3*—supporting pillar, *4*—burner, *5*—filling screw, *6*—switch, *7*—spiral, *8*—brass cap, *9*—contact pin, *10*—contact tube, *11*—fuel container, *12*—battery compartment

A standard safety lamp top of approved design screws on the fuel container, with a magnetic lock, designed to make a perfect seal with the body of the indicator The greatest care is taken in manufacturing and setting the spiral spring Every Spiralarm works under rigidly controlled test conditions before leaving the factory. The final setting is made in mixtures of methane and air.

Operating principle. As issued from the lamp room the Spiralarm flame burns steadily at the height indicated by pointer *2*. The heat generated during the warming-up period has caused the thermostatic spiral to unwind until the contact pin *9* is within 4.8 mm of the contact tube *10*. Under normal air conditions the height of the flame remains constant irrespective of the temperature of the surrounding air.

If the air contains 1.25 or 2.5 per cent of gas, the flame increases in size and produces greater heat which, in turn, causes the spiral to unwind still farther. The contact pin passes through the slot in

The methane signalling and control apparatus consists of automatic elements, including an electric supply for the detector and an interlocking switch housed in a welded metal box. In the upper part are the methane indicator and alarm siren. The apparatus is 850 mm wide and 800 mm high.

The signal board is furnished with a warning siren and is contained in a metal box with the secondary relays for signalling and the signal lamps inside it. One board can serve five detectors.

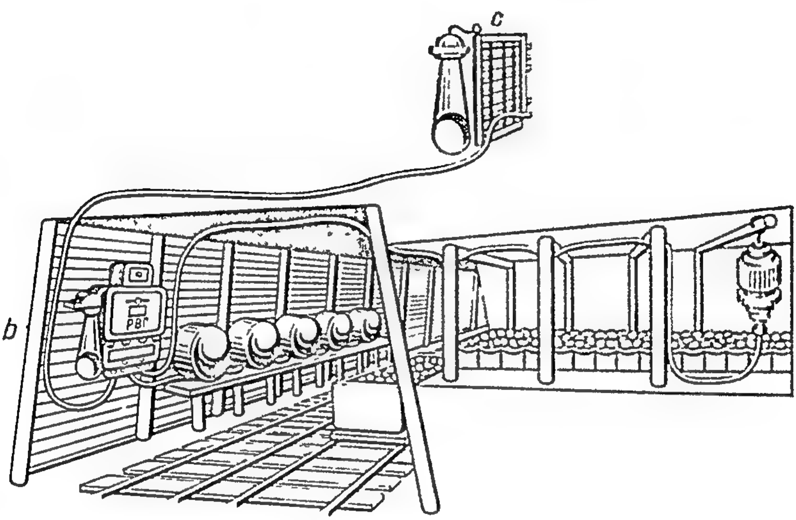

Fig. 1-15 Automatic methane detector, type ИМ-2

On the lid of the board are 20 lamps with tinted lenses (four lamps for every detector) indicating the operation of the instrument. The accuracy of measurement and of response of the whole unit is 0.2% methane.

The stationary automatic methane detector type ИМ-3 differs from the ИМ-2 instrument in that it does not require a special cable and consumes the high-frequency current from telephone cables.

The *stationary thermocatalytic methane relay* type АМТ-2 is used for continuous automatic monitoring of the methane content in exhaust air. The instrument gives alarm signalling and shuts off electrical power to face equipment when the methane content in the air exceeds a safe limit. The operation of the methane relay is based on the flameless burning of methane on the surface of a carrier catalyst at a burning temperature of 360°C.

In mines in Britain and other countries, Type "M" Spiralarms are widely used. Subjected to every condition of regular service, they have been absolutely satisfactory in every way.

1-9. PRINCIPAL METHODS OF PREVENTING UNDERGROUND IGNITIONS OF METHANE

Until very recently the principal method of preventing methane underground has always been ventilation, that is, the maximum dilution of the explosive mixture with intake air, preferably to one-fifth or one-tenth of the lower limit of explosibility, with removal of the diluted gas out of the mine to the surface.

The principle of this method is highly imperfect because the explosive gas passes out of the mine through a long circuit of underground roadways, and dangerous concentrations of gas in production faces or old workings are not prevented

It would be more rational somehow to render the explosive gas harmless at the point of emission, but this is extremely difficult because methane ordinarily is very stable

Around 1938 the search began for some inert gas of high heat capacity which if added to methane would dilute it to a non-explosive mixture in air Laboratory experiments showed that dilution of 1 volume of methane with 6 volumes of nitrogen, or 3 3 volumes of carbon dioxide, or 1.4 volumes of such gases as carbon tetrachloride (CCl_4) or dichlorodifluoromethane (CCl_2F_2) will make methane non-inflammable These experiments have not so far given positive results because to prevent methane explosions in this way (even if only one volume of the neutralizing gas is added for one volume of explosive gas), a vast daily consumption of neutralizers would be needed and this is not preferable from the cost standpoint.

The All-Union Coal Research Institute has investigated in the laboratory the neutralizing of methane underground by bacteria which oxidize methane to carbon dioxide Verifying experiments did show that these diluted bacterial cultures, obtained from liquid manure or the setting tanks of biological stations, do in fact devour methane actively at a temperature of about 30° C, but the process is slow (see Table 1-9) To solve this problem of the practical use of bacte-

TABLE 1-9. Change in Composition of Mixture of O_2 and CH_4 Held for 312 Hours in a Glass Flask. Original Volume of Mixture — 900 cm³, Final Volume — 300 cm³. (Experiments by the All-Union Coal Research Institute, 1938)

Component	Original mixture		Final mixture	
	per cent	cm³	per cent	cm³
CH_4	15.5	145.0	1 74	5.0
O_2	58 0	520 0	49 5	148 0
CO_2	2 2	20.0	10 2	31 0
CO	1 8	16.0	0 85	2.5
H_2	1 1	10 0	0 0	0 0

Note The absolute quantity of methane oxidized was 140 cm³, or 97 per cent

the contact tube and presses together the two internal contacts, completing the alarm lamp circuit and switching on the red alarm, which continues to show "danger" until the gas content is lowered or the detector is removed from the gassy area. The time required for operation is only three minutes from the first appearance of gas in sufficient concentration (1 25 or 2.5 per cent).

Before the Spiralarms are taken underground they are checked. This is done as follows. after the Spiralarms have been lit and allowed to warm up for one hour, they are placed—in batches of six—in a testing cabinet containing a mixture of fuel and air in the correct proportion

In 100 hours of burning the lamp uses 280 grams of high-grade paraffin oil.

The Spiralarm settings are 1.25 or 2.5 per cent of gas. It weighs in all 2 kg and is 30 cm high.

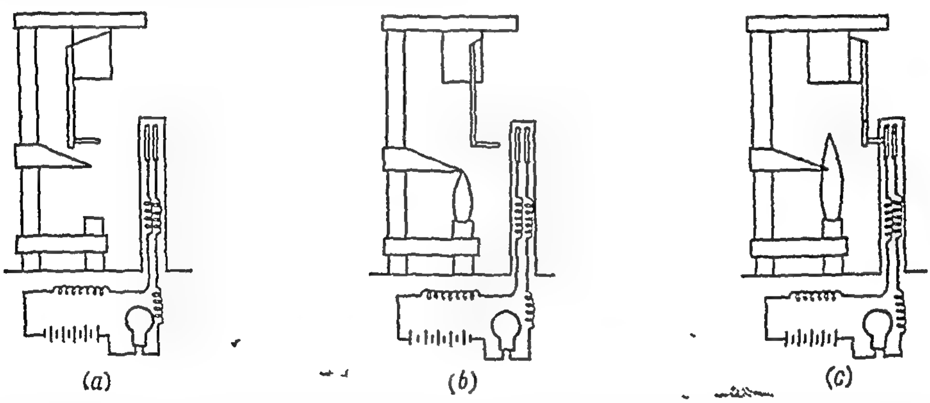

Fig. 1-17 Operation of type "M" Spiralarm detector

a—position of spiral and contact pin, Spiralarm cold, *b*—position of spiral and contact pin with a normal flame, *c*—alarm position of spiral and contact pin, with the larger flame, the pin closes the lamp contacts

The Ringrose pocket gas analyser gives a rapid indication of the gas content in the mine air. It functions on the principle of using the property of gases to pass through a porous membrane at different speeds, enabling them to separate out of a gas mixture. The Ringrose analyser uses this property to feed methane into a combustion chamber containing a heated filament. The methane diffuses faster through the porous wall of the combustion chamber than the other components of the mine air, meeting the hot filament and burning. The water vapour thus formed condenses, reducing the pressure in the combustion chamber proportionately to the quantity of gas which has entered the chamber. This pressure reduction is determined by a special spirit manometer which is graduated to read directly in methane percentages.

air stream In very gassy faces the air may be supplied to the face by brattices or ventilation ducting.

4. The mine ventilation as a whole and that in the working districts should be the simplest, most reliable and least liable to damage.

5 The mine should be ventilated by exhaust ventilation, while dead ends must be ventilated mainly by forced ventilation if an auxiliary fan is used

6. Ascensional ventilation should be used in inclined faces when the angle of dip is 5° or more.*

7. Faces should be laid out in sub-horizons and horizons with the lower faces advancing ahead of the upper ones, a layout like overhand stoping.

8. The faces should, where possible, be ventilated by the mine head. Where auxiliary fans are used they must be installed in such a way that they receive intake air, and the fan inlet must therefore be at least 10 metres away from the point where the air returns from the face

9. Ventilating doors should be rationally laid out and reliably operated.

10. The method of working should be such as to ventilate the face easily and reliably with a minimum number of dead ends, in particular, to the rise.

11. Leakages of air must be minimized.

12. The ventilation and the gas emission must be watched, with systematic measurements of gas and of the air flow through the workings, for this it is necessary to have:

(a) systematic additional ventilation plans and regularly kept log books of ventilation and gas measurements;

(b) systematic air sampling;

(c) instruments for checking the ventilation (anemometers, deprimometers, water gauges) and gas detectors;

(d) a special staff for ventilation and gas measurement.

Air samples are taken not less than two to three times per month depending on the gas emission from the mine. Methane measurements must be made in every working area, including engine rooms, before the beginning of every shift, not less than every 1-3 hours during the shift, depending on the category of the mine, and one hour before the end of the shift.

---

* Recently, the requirement of ascensional ventilation for inclined production faces in gassy mines has come to be regarded as controversial because even at an air velocity of 1-1 5 m/sec the stream is in steady turbulent flow

In these flow conditions it is practically impossible for the methane to layer and accumulate above the air under the roof, moreover in dusty or hot mines, descensional ventilation has a number of significant advantages.

ria, further work is needed on the cultivation of more active bacteria of this type, including a search for bio-catalysts to accelerate the bacterial neutralization of methane, and the starting of experiments in conditions approaching those underground

The whole complex of the prevention of methane ignitions underground can be divided into the following groups:

1-9.1 Prevention of Dangerous Accumulations of Gas Underground

The methane-air mixture must be diluted in productive workings to a concentration less than or equal to 0 5-1%, and in the general return airway of the mine down to less than 0 75% by volume.

To achieve this a continuous supply of pure air must be sent down the mine, corresponding in volume to the amount of the gas emission; the air must be distributed throughout the mine in the network of airways so as to remove the impure air continuously to the surface, to achieve this the following requirements should be observed

1 The mine should be ventilated by mechanical fans, and not by natural ventilation. The main fan installation must consist of two independent fans with separate drives, each designed to pass the full volume of air required by the mine. For mines which are not gassy, ventilation by a single main fan is allowed, with two drives, including the stand-by. The fans must be provided with the layout for reversing the air current and the power supply must come by two distinct power lines from a power station or substation.

Mines of Category 1 are allowed to have a reserve feeder for the fan power supply from a secondary electrical distribution point.

If the main fan stops, the men must immediately be removed from the faces into the intake airway and the power supplies must be cut off from the districts, but if the stoppage lasts more than 30 minutes the men must go to the shaft After a fan stoppage the power supply to the working districts may not be switched on nor may work in them start until the workings have been ventilated and the gas has been removed.

2 The equivalent orifice of the mine should be large enough, larger than 1 5 m² (further details on equivalent orifices are given in Part Two, Chapter 8) at a low water gauge depression, preferably not more than 150-300 mm of water gauge.

3. The faces should be ventilated by an active air stream and not by diffusion Diffusion is allowed only in dead ends if they are not gassy, for a maximum distance of 10 metres from the active

which receives intake air from the main intake airway at the given horizon or wing and discharges its return air to the main return airway of the same horizon or wing.

The return air from a face in a seam must not be directed to any other place in any seam at any other horizon.

Under the Safety Regulations, mines which are dusty or belong to gas Categories 2 or 3, or are outside the categories, must be ventilated in such a way that every face together with its development roads is ventilated by its own air current. In mines of gas Category 1, series ventilation of two faces on one horizon is permitted in seams which are neither dusty nor subject to spontaneous combustion provided that their total length does not exceed 320 m, that the distance between them is not more than 200 m, and that the return air from the first face does not contain more than 0.5% of methane

2 The layout of the main intake and return airways of a wing or panel incline to the rise or the dip, should be such that no explosion can short-circuit the ventilation.

It is dangerous for the main intake and the main return airways to be close to each other and separated only by a narrow pillar cut through by stentons even though these stentons may be stopped with walls.

3. Suppression of dust ignitions and their propagation through the mine (Chapter 2).

4. There must be a permanent and clearly understood rescue service. The emergency mine-recovery plan must be well thought out and known to the technical staff and the miners, the same applies to refuge rooms and supplies for the rescue workers.

5. The miners must know the properties of firedamp, the precautions to be taken with it, and the necessary action for self-rescue and for rescuing others.

1-10. REMOVAL OF GAS FROM COAL SEAMS

The degassing or firedamp drainage of coal seams is a process of extraction of the firedamp from the seams, veinlets and neighbouring rocks and from the wastes. It includes the isolated removal of the methane-air mixture to the surface. The purpose of firedamp drainage is the lowering of the methane emission in the workings.

Methane drainage on a large scale was first employed in the Ruhr district of Germany in 1943. Since then it has been widely used in all the main coal-producing countries and in fact has revolutionized not only methods of reducing mine gas but also the techniques of working gassy coal deposits. The first firedamp drainage installation in the USSR was commissioned in 1951. At present these

With gas contents above 1%, action must be taken to remove the accumulation of gas, and at 2% the men must leave the gassy district.

1-9.2 Prevention of the Ignition of Methane Accumulations

The measures indicated above are aimed at eliminating explosive concentrations of methane underground. However, this, so to speak, first line of defence against methane ignitions is inadequate; there must be a second line of defence aimed at preventing ignitions of methane underground. The measures included in the second line of defence are as follows:

1. It is forbidden to have naked lights or matches underground and to smoke.
2. Battery lamps should be used for lighting; oil lamps with two gauzes and a metal bonnet or special gas analysers should be used for gas testing.
3. Shotfiring must be minimized, and replaced by hydraulic breaking of coal, and heavy pneumatic picks must be used for the ripping of the roof in roadways
4. During shotfiring:

(a) only permissible explosives and permitted accessories must be used;

(b) electric instantaneous or short-delay detonators must be used with a delay according to current Safety Regulations;

(c) holes must be properly stemmed; the stemming must be prepared from incombustible, friable or plastic materials (stone, dust, sand or clay) and must occupy not less than one-third of the length of the hole. No shots may be fired outside a stemmed shothole, no shothole may be shorter than 65 cm, and no stemming may be shorter than 50 cm.

Shotfiring is allowed only when the methane content is less than 1% at the face and for 20 metres away from it. If the coal face is dusty, the face and the roadway for 20 metres away from it must be stone-dusted.

5. The precautions needed in the use of electric current must be observed; the use of explosion-proof electrical equipment is required.

With very heavy gas emissions, electrical equipment should be exchanged for equipment driven by compressed air.

1-9.3 Main Measures for Localizing Gas Ignitions Underground

1. The mine is divided into the largest possible number of independent ventilating districts, an independent district being one

ing the developments (haulage roads, inclines, etc.) the boreholes can be drilled in the seam:

(a) from "stables" on both sides of the roadway at an angle of 15-20° and 10-15 metres long, during driving;

(b) on both sides of the roadway at a distance of about 10 m from it throughout the length of the roadway (during the driving of inclines, for example).

During coal face production the boreholes are drilled in the seam ahead of the face at 10-15 m from each other for the full height of the horizon to a small distance from the other road. The holes are drilled from the lower or the upper road. During drilling, the holes usually give very little gas, but as the face approaches, or the holes enter the area in which the effect of the face work is felt, they begin to give off gas.

Another method of extracting gas from a seam being mined is as follows: part of the mine take is cut up by development roads into fairly small rectangles, the developments then being blocked up, after the stoppings have been built, the pipes passing through them are connected to vacuum pumps at the surface. This part of the coal seam is drained for six months or more, the vacuum pumps sucking the gas from the area. The production faces are then started. This method is expected to reduce the gas emission by 25-30%.

Methane drainage of neighbouring seams is the most widespread technique used for removing the gas from coal seams. It consists in drilling boreholes in the roof (rarely the floor) from the intake or the return airway until the neighbouring seams are intersected, which are in the roof at a distance of 70-90 m or in the floor up to 25-30 m away. The boreholes are drilled sloping towards the waste; the hole diameter varies from 50 to 85 mm. The collar of the hole is sealed for a length of 4-6 m in solid ground and 7-10 m in weak or cracked ground; the sealed length of the hole is reamed out to a diameter of 120-150 mm and a pipe is cemented in it, through which the gas is led out. All the holes are equipped with water separators and monitoring devices for measuring the gas flow and the pressure, and to allow gas samples to be taken for determining the methane content.

Interesting results were obtained in West Germany by using long casings of 21-30 m. Introduction of these, instead of the former casings 10 m long, increased the gas flow from an average 1,400 m^3 per day to 5,780 m^3; the suction in the borehole was 1,430 mm water gauge. The spacing between boreholes can thus be tripled or quadrupled.

In the Donets Basin (gently sloping, fairly thin seams) the angle β of inclination of the boreholes from the return airway is deter-

nstallations are working in more than 100 mines. From all the Soviet mines about 1,100 million m^3 of methane are now extracted every year.

The advantages of firedamp drainage include:

1 Reduction of the quantity of methane released into the mine, significantly improving the safety of working.

2 Reduction of the gas quantity liberated allows the volume of underground work spent on diluting the gas to be greatly reduced which in turn

(a) enables the cross section of the mine airways to be reduced;

(b) reduces ventilation costs,

(c) allows faces to be lengthened and in this way often enables the horizon to be worked for its full length and not split into two sub-horizons.

3. Increase of coal output and speed of advance of coal faces because of the reduction of idle times due to excess of gas

4 The possibility of using a large quantity of gas of high calorific value.

According to current theories, the gas from the neighbouring seams is emitted into the workings in the following way If a seam is underworked or overworked by another seam, the rock pressure on it diminishes Partial unloading from the coal of the rock pressure and accompanying tensile deformations with the opening of cleavage and other cracks, abruptly increase the gas emission Movement of the rock in the roof over the waste is most intensive directly over the seam being worked High rates of displacement favour the development of heavy cracking in this part of the mass As the distance from the face increases, the rate of movement of the ground diminishes (smooth lowering) and consequently the development of cracking also diminishes. The hollows formed by bed separation and cracking form reservoirs in which gas can accumulate, to be emitted later into the workings.

As the faces advance, the intensity of gas emission from the bore holes at first increases and then gradually diminishes The second underworking or overworking leads to some renewal of gas emission from boreholes which have stopped emitting gas.

Methods of removal of gas Degassing can be from:

(a) the seam being worked;

(b) neighbouring workable or unworkable seams;

(c) the worked-out area.

Firedamp drainage from a seam while it is being worked is now in the stage of investigation, development, and partial introduction; it is used mainly in those mines where natural conditions forbid the firedamp drainage of neighbouring seams Boreholes are drilled in the seam being worked and the gas is sucked out of them In driv-

ing the developments (haulage roads, inclines, etc.) the boreholes can be drilled in the seam:

(a) from "stables" on both sides of the roadway at an angle of 15-20° and 10-15 metres long, during driving;

(b) on both sides of the roadway at a distance of about 10 m from it throughout the length of the roadway (during the driving of inclines, for example).

During coal face production the boreholes are drilled in the seam ahead of the face at 10-15 m from each other for the full height of the horizon to a small distance from the other road. The holes are drilled from the lower or the upper road. During drilling, the holes usually give very little gas, but as the face approaches, or the holes enter the area in which the effect of the face work is felt, they begin to give off gas.

Another method of extracting gas from a seam being mined is as follows: part of the mine take is cut up by development roads into fairly small rectangles, the developments then being blocked up, after the stoppings have been built, the pipes passing through them are connected to vacuum pumps at the surface. This part of the coal seam is drained for six months or more, the vacuum pumps sucking the gas from the area. The production faces are then started. This method is expected to reduce the gas emission by 25-30%.

Methane drainage of neighbouring seams is the most widespread technique used for removing the gas from coal seams. It consists in drilling boreholes in the roof (rarely the floor) from the intake or the return airway until the neighbouring seams are intersected, which are in the roof at a distance of 70-90 m or in the floor up to 25-30 m away. The boreholes are drilled sloping towards the waste; the hole diameter varies from 50 to 85 mm. The collar of the hole is sealed for a length of 4-6 m in solid ground and 7-10 m in weak or cracked ground; the sealed length of the hole is reamed out to a diameter of 120-150 mm and a pipe is cemented in it, through which the gas is led out. All the holes are equipped with water separators and monitoring devices for measuring the gas flow and the pressure, and to allow gas samples to be taken for determining the methane content.

Interesting results were obtained in West Germany by using long casings of 21-30 m. Introduction of these, instead of the former casings 10 m long, increased the gas flow from an average 1,400 m^3 per day to 5,780 m^3; the suction in the borehole was 1,430 mm water gauge. The spacing between boreholes can thus be tripled or quadrupled.

In the Donets Basin (gently sloping, fairly thin seams) the angle β of inclination of the boreholes from the return airway is deter-

mined by the equation·

$$\beta = \arc\tan \frac{N}{N \cot (\gamma + \alpha) + C} - \alpha \qquad (1\text{-}4)$$

where N = perpendicular distance from the seam being mined to the seam being drained, metres

α = angle of dip of the seam

C = distance equal to the sum of the widths of the coal pillar above the return airway and the roadside pack below it, plus the roadway (3 m) and an additional 5 m

γ = angle which, depending on the distance N, is equal to:

70° with $N = 3\text{-}10$ m
75° with $N = 10\text{-}30$ m
80° with $N = 30\text{-}80$ m

If the holes are drilled from the haulage road, the sign of the angle α changes.

The length of the holes is obtained from the formula

$$L = \frac{N}{\sin (\beta + \alpha)} \text{ metres} \qquad (1\text{-}5)$$

using the same notation as before.

The spacing between the methane drainage holes must be taken from Table 1-10.

TABLE 1-10 Recommended Spacing of Holes Drilled for Firedamp Drainage

Perpendicular distance from seam being mined to adjacent seam, metres	Spacing of holes drilled for firedamp drainage, metres	
	in hard coal measures	in soft coal measures
From 40 to 60 times the thickness of the seam being mined	150-200	100-150
From 20 to 40 times the seam thickness	80-150	40-80
From 5 to 20 times the seam thickness	40-50	30

The gas is sucked out by rotary vacuum pumps of PMK type (Models 2, 3, 4 and 5) with an output of 4.2 to 50 m^3/min. Usually the pumps are installed on the surface. The gas pipe diameter is

from 150 mm for the district pipes to 300 mm for the mains and the shaft pipes.

The methane content in the gas removed depends on the efficiency of the seal at the collar of the hole and on its length; long boreholes usually give richer gas: with good sealing it is possible to recover the gas with a methane content of up to 80% or more; but air sucked in can reduce the methane content to 25-30%. The flow from one borehole varies widely from 0.5-3 up to 10-15 m^3/min. The operating period of a hole generally does not exceed 6-10 months, but some boreholes have operated for a year or more.

Firedamp drainage reduces the gas emission in the districts and for the mine as a whole; in favourable conditions it is possible to lower the gas emission from the districts by 60-75% or even more and for the mine as a whole by 35-45%.

The minimum gas emission at which it is expedient to begin firedamp drainage depends on a number of technical and cost factors: the costs of the drainage work, of driving the roads, of power, etc. Some very gassy deposits are never worked without methane drainage. At present a gas emission of 20-25 m^3 per ton is thought to be the level at which firedamp drainage should start.

To ensure safety during the firedamp drainage work the following should be observed:

(a) An active air flow must be ensured to the drilling rigs underground, and during the subsequent period of drawing off the gas through boreholes the methane content must be carefully watched.

(b) No explosive mixture must be allowed in the suction pipes due to air sucked into them through leaky joints.

(c) On the surface in the vacuum pump room, all the safety rules must be strictly observed and the readings of the monitoring and recording instruments must be carefully watched. All the equipment in the pump room must be flameproof. To prevent gas explosions the equipment installed must record: the value of the gas vacuum on the suction side; the value of the gas pressure on the discharge side; the methane content in the gas sucked out (not less than 30% in the USSR and not less than 40-50% in other countries); the gas flow, the gas temperature, the temperature of the body of the air blower; the calorific value of the gas, which should be not less than 2,800 kcal/m^3.

To increase the release of gas from coal seams, present practice is as follows:

(a) Chambering the drainage holes so as to increase their gas output; a "torpedo" carrying 7.5 kg of explosive is pushed into the end of the borehole. After chambering, the gas emission increases by 30%.

(b) By the so-called hydraulic disintegration of the seam; boreholes are drilled from the surface until they intersect the seam; water is then pumped into them at a pressure of several hundred atmospheres, followed by sand which holds open the cracks formed in the seam, from these holes which are about 400 m deep a considerable amount of gas has been obtained in the Karaganda coalfield.

A special method of firedamp drainage of neighbouring seams is used in the Saar coalfield. in an unworked seam in the roof of the worked seam a road is driven, from which boreholes are drilled upwards, downwards and in the unworked seam, the roadway is closed with stoppings and methane is sucked out of it, a method known as Hirschbach's method.

The last method of firedamp drainage is the removal of gas from the waste; the following variations of this technique are used:

(a) Pipes are laid in the waste and joined to the gas suction main in the return airway, the pipes being passed through the strip packs.

(b) Holes are drilled into the waste from a neighbouring seam and the gas is sucked out through the boreholes.

(c) Holes are drilled into the caved area of a seam from the surface or from an overlying horizon and the gas accumulated in this area is removed.

The quantities of gas obtained in the various methods of firedamp drainage are given below, according to G.D. Lidin:

(a) Firedamp drainage by boreholes in neighbouring seams	54 5%
(b) Ditto, Hirschbach's method	4 2%
(c) Drainage of wastes	13.4%
(d) Drainage of the seams being worked	10.2%
(e) Drainage by boreholes during the driving of a roadway	8 2%
(f) Drainage by boreholes from the surface and special vertical shafts	8.6%
(g) Chambering, or hydraulic disintegration of the seam	0 9%

1-11. PREVENTION OF METHANE IGNITIONS AT COAL STOCKS ON THE MINE SURFACE

Methane is emitted from the coal for a long time even after the coal has reached the surface, flashes or even explosions of the gas in bunkers or other enclosed spaces have occurred because of the high intensity of gas emission from the coal being stored. To prevent ignitions the storage places must be well ventilated and entry to them must be allowed only with safety lamps.

CHAPTER 2

MINE DUST

2-1 GENERAL

Mine dust consists of small to minute particles of mineral and rock, from 1 mm in size down to a fraction of a micron, suspended in the mine air or settled on the walls, floor and roof of the workings.

The ability of dust particles to remain suspended in the air for some time depends on their fineness, shape, and density, as well as on the humidity, temperature and velocity of the air flow.

Theoretical calculations and experiments have shown that solid spherical particles with a density of 2.5 (quartz), falling freely in still air have the following speeds:

Particle mm	diameter microns	Speed
0 1	100	78 6 cm/sec
0 01	10	7.86 mm/sec
0 001	1	0 0786 mm/sec
0 0001	0 1	0 000786 mm/sec

From this it follows that in still air, dust particles of known density and shape which are at any moment at one metre from the floor, will settle on to it during the time indicated in Table 2-1.

TABLE 2-1 Mine Dust

Diameter, microns	100	10	5	1	0 5	0.2
Settling time	1 3 sec	2.2 min	9 min	3 hrs	11 hrs	46 hrs

Mine dust is undesirable from two points of view (a) as a health hazard, and (b) as a possible cause of explosion

The unhealthy airborne dust is smaller than 5-10 microns, usually about 1-2 microns in size but that causing explosions can include grains up to 1 mm, whether airborne or settled.

A more accurate impression of fine dusts is given by their classification, gaseous dispersed systems (aerosols) being divided into the following three classes according to their particle dimensions:

1. Dust proper—aerosols with particle dimensions above 10 μ (these fall in still air with increasing speed).
2. "Fog"*—particles from 10-0.1 μ (these settle in still air with a constant speed according to Stokes' law).
3. "Smoke", particles of 0.1 μ and smaller which resemble gas molecules by their behaviour in air; they are in continuous thermal motion, are rapidly dispersed and do not settle at all.

2-2. MINE DUST AS A HEALTH HAZARD

1. The dustiness of the air, that is the quantity of dust contained in it, is described in two ways:

(1) as the number of mg of dust per m^3 of air, the weight or gravimetric method;

(2) as the number of dust particles per cm^3, the dust count method.

Not only mine air but all air on the earth's surface contains dust. According to some data, even the air on high mountains may contain 200 to 1,000 dust particles per cm^3 and in cities as many as 50,000 to 200,000 particles. According to many observations the weight of dust suspended in the air is:

Dwellings	about 1.5 mg/m^3
Sawmills	about 15-17 mg/m^3
Crushing mills	22-47 mg/m^3
Cement works and ore treatment works	130-200 mg/m^3

the first figure expressing the level during interruptions in work.

One reason for mine air pollution, apart from the fact that mineral is broken and dug, may be that there are rock tips and mineral tailings dumps at the surface not far from the shafts. Thus, for example, a test of the air around the tip of a mineral preparation works at one of the metal ore mines of the Altai indicated 20 mg/m^3 of dust particles smaller than 5 μ across. The dust in mine air can be caused by the proximity of picking tables for ore or coal as well as by low factory chimneys nearby.

When the dust is not poisonous we can consider that:

at dust contents less than 1 mg/m^3 the air is not dusty
ditto 5 mg/m^3 ditto moderately dusty
ditto 10 mg/m^3 ditto dusty
ditto 20 mg/m^3 ditto very dusty
ditto 100 mg/m^3 ditto extremely dusty

* Fog is also the name given to aerosols with a dispersed phase consisting of liquid particles

For conversion of weight standards to dust count standards it is accepted that 1 mg/m^3 corresponds to about 200 dust particles (up to 2 μ across) per cm^3. As far as mine air is concerned, the quantity of airborne dust in active mine workings fluctuates between several mg and several hundred mg/m^3, reaching at times some grams (near chutes, in faces with machine-cut coal, during the working of cutter-loaders, etc.), for example, 5 to 7 or even 10 to 15 g/m^3.

2. According to its action on man's health, dust can be divided into two classes: (1) poisonous dust (lead, mercury, arsenous, etc); and (2) non-poisonous dust. (a) vegetable substances (cotton, hemp), and (b) non-poisonous mineral substances, such as coal, ore, and various rocks

Even non-poisonous dust is harmful if the air contains much of it, because it irritates the eyes and respiratory tract and affects the lungs, and thus disturbs the functions of the human body as a whole, causing illnesses with the general name of *pneumoconiosis*, from the Greek words "pneuma" for breathing and "konios" for dust.

The maximum permissible concentrations of non-poisonous airborne dusts in industrial working areas are given in Table 2-2.

TABLE 2-2 **Permissible Concentrations of Non-Toxic Mine Dust**

Type of dust	Permissible concentration, mg/m^3
Dust containing more than 70% free silica	1 0
Dust containing 10 to 70% free silica	2.0
Coal or coal-measure dust containing more than 10% free SiO_2	2 0
Coal dust containing less than 10% free silica	4 0
Coal dust containing no free silica	10.0

A particularly severe form of pneumoconiosis accompanied by consolidation of the lungs (fibrous degeneration of the lung tissue) with the formation of characteristic tubercles of consolidated tissue is caused by silica dust and is called *silicosis* (Latin *silicium* = flint). Lung disease caused by coal dust is called *anthracosis* (Greek *anthrakon* = coal); that caused by the action of asbestos dust is called *asbestosis*.

A more accurate impression of fine dusts is given by their classification, gaseous dispersed systems (aerosols) being divided into the following three classes according to their particle dimensions:

1. Dust proper—aerosols with particle dimensions above 10 μ (these fall in still air with increasing speed).

2. "Fog"*—particles from 10-0.1 μ (these settle in still air with a constant speed according to Stokes' law).

3 "Smoke", particles of 0.1 μ and smaller which resemble gas molecules by their behaviour in air; they are in continuous thermal motion, are rapidly dispersed and do not settle at all.

2-2. MINE DUST AS A HEALTH HAZARD

1. The dustiness of the air, that is the quantity of dust contained in it, is described in two ways:

(1) as the number of mg of dust per m³ of air, the weight or gravimetric method;

(2) as the number of dust particles per cm³, the dust count method.

Not only mine air but all air on the earth's surface contains dust. According to some data, even the air on high mountains may contain 200 to 1,000 dust particles per cm³ and in cities as many as 50,000 to 200,000 particles. According to many observations the weight of dust suspended in the air is:

Dwellings	about 1.5 mg/m³
Sawmills	about 15-17 mg/m³
Crushing mills	22-47 mg/m³
Cement works and ore treatment works	130-200 mg/m³

the first figure expressing the level during interruptions in work

One reason for mine air pollution, apart from the fact that mineral is broken and dug, may be that there are rock tips and mineral tailings dumps at the surface not far from the shafts Thus, for example, a test of the air around the tip of a mineral preparation works at one of the metal ore mines of the Altai indicated 20 mg/m³ of dust particles smaller than 5 μ across. The dirt in mine air can be caused by the proximity of picking tables for ore or coal as well as by low factory chimneys nearby.

When the dust is not poisonous we can consider that·

at dust contents less than 1 mg/m³ the air is not dusty
ditto 5 mg/m³ ditto moderately dusty
ditto 10 mg/m³ ditto dusty
ditto 20 mg/m³ ditto very dusty
ditto 100 mg/m³ ditto extremely dusty

* Fog is also the name given to aerosols with a dispersed phase consisting of liquid particles

For conversion of weight standards to dust count standards it is accepted that 1 mg/m^3 corresponds to about 200 dust particles (up to 2 μ across) per cm^3. As far as mine air is concerned, the quantity of airborne dust in active mine workings fluctuates between several mg and several hundred mg/m^3, reaching at times some grams (near chutes, in faces with machine-cut coal, during the working of cutter-loaders, etc.), for example, 5 to 7 or even 10 to 15 g/m^3.

2. According to its action on man's health, dust can be divided into two classes. (1) poisonous dust (lead, mercury, arsenous, etc.); and (2) non-poisonous dust. (a) vegetable substances (cotton, hemp), and (b) non-poisonous mineral substances, such as coal, ore, and various rocks

Even non-poisonous dust is harmful if the air contains much of it, because it irritates the eyes and respiratory tract and affects the lungs, and thus disturbs the functions of the human body as a whole, causing illnesses with the general name of *pneumoconiosis*, from the Greek words "pneuma' for breathing and "konios" for dust.

The maximum permissible concentrations of non-poisonous airborne dusts in industrial working areas are given in Table 2-2.

TABLE 2-2 Permissible Concentrations of Non-Toxic Mine Dust

Type of dust	Permissible concentration, mg/m^3
Dust containing more than 70% free silica	1 0
Dust containing 10 to 70% free silica	2 0
Coal or coal-measure dust containing more than 10% free SiO_2	2.0
Coal dust containing less than 10% free silica	4 0
Coal dust containing no free silica	10 0

A particularly severe form of pneumoconiosis accompanied by consolidation of the lungs (fibrous degeneration of the lung tissue) with the formation of characteristic tubercles of consolidated tissue is caused by silica dust and is called *silicosis* (Latin *silicium* = flint). Lung disease caused by coal dust is called *anthracosis* (Greek *anthrakon* = coal); that caused by the action of asbestos dust is called *asbestosis*.

Men working in dusty air often develop acute silicosis after 5-10 years but the illness has been known to appear after only 1.5-3 years.

Of the dust inhaled by man, a considerable part goes no further than his nose and throat and is thrown out again, the remainder (usually the smallest particles, below ten microns in cross section) reaches the lung alveoli where a large part of it stays, irritating and consolidating them (fibrosis), some of it is dissolved and goes into the blood. Thus, for example, it has been proved that dust particles of silica are partly transformed in the alveoli into poisonous silicic acid (H_2SiO_3) which passes into the blood.

Post-mortem examinations of individuals who have died of pneumoconiosis sometimes revealed extremely large concentrations of mineral dust in the lungs, up to 9 weight per cent of the lungs.

The pathological action of rock dust on the lung alveoli is exceedingly complicated and has not yet been adequately explained, there is therefore no known means of curing silicosis, and emphasis is laid on maintaining the resistance of the organism, on respiratory gymnastics, and some other means of curing which are effective only in the early stage of the disease.

There are two points of view on the main cause of silicosis: some scientists say it is the dispersion of the dust, that is its size distribution, and in particular its content of the smallest fraction below 5 microns, others that it is the mass (weight) of the dust. In the Anglo-Saxon countries it is believed that the finest dust is the most dangerous and therefore the permissible amounts of dust in mine air are expressed as the number of dust particles smaller than 5 microns per cu cm. In other countries, including the USSR, it is believed that a more correct indication of the harm done by dusty air is its weight per unit of volume. The permissible contents of various types of dust are given in Table 2-2.

The weight standards for dust are based on the work of E. V. Khukhrin, which was carried out in two directions. In one test series quartz dust was introduced into the lungs of animals, the dust being uniform in weight and mass but of varying particle size. In the second series of experiments the dust introduced into the animals' lungs contained the same number of particles but they were of different diameters. The action of six sizes of particles was investigated: up to 1 μ; 2 μ, 3 to 5 μ, 5 to 10 μ, 10 to 20 μ, 20 to 40 μ. Parallel investigations were carried out into the action of ordinary quartz dust of roughly uniform size distribution. The time required for the onset of silicosis was studied, its severity, and the life span of the animals.

The first series of experiments showed that silicosis may be caused by the action of all the dust sizes—very fine, coarse and the ordinary

dust. With the same mass of dust inhaled, silicosis starts more quickly from dust of size 2 to 3 microns, and more slowly from coarser dust of 5 to 10 microns and from ordinary dust.

In the second series of experiments, equal numbers of particles, about 1,500,000 of different coarseness and mass were introduced into the animals.

About $1^1/_2$ million particles of the coarsest dust of 20 to 40 microns weighed 50 mg; particles of 10 to 20 microns weighed 6 mg; those of 5 to 10 microns weighed 1 mg; those of 3 to 5 microns weighed 0.1 mg; those of 2 microns weighed 0.01 mg, and those below 1 micron weighed 0.001 mg.

In this series of experiments only the very coarsest dust, weighing 50 mg clearly resulted in the development of silicosis. The dust of 10 to 20 microns, with a mass of 6 mg gave only a very slight reaction in the proliferation of cell elements in the lymphatic follicles of the lungs, a slight thickening of the interalveolar septa, and isolated silicotic nodules in the lymph nodes All the remaining finer dust fractions caused no morphological changes in the lungs Tests have shown that fine dust smaller than 2 microns can initiate silicosis only when its total mass is large. If the mass of this fine dust is small, the body can effectively combat it by its own protective mechanisms. These experiments on animals thus showed that the main factor in the initiation of silicosis is the mass of the active substance.

It is known that in any mine and industrial dust the weight of the fine dust is always very small. The main mass of dust is in the particles larger than 3 to 5 microns, and they weigh considerably more than the fine dust. It is therefore natural that the course of a silicosis should be determined by the ratio between the masses of the fine and the coarse dust.

This exceedingly complicated and important problem of the biological significance of the mass and the number of the dust particles needs further careful study with various types and doses of dust.

It has been established that the most harmful dust is either entirely or largely free silica; therefore the criterion for the harmfulness of non-poisonous mineral dust is the content of silica dust

The most harmful is siliceous, quartz, sandstone and granite dust. The less harmful silicate dust includes feldspar and talc, even less harmful is limestone, shale, and clay dust. Health standards for Soviet mines indicate that dust with a free silica content above 10% is so harmful that the permitted content of silica dust in the mine air should not exceed 2 mg/m^3. Coal dust, generally speaking, is one of the least harmful but by the same standards the content of coal dust in the air should not exceed 10 mg/m^3.

The worst sufferers from dust are coalcutter and cutter-loader operators, drillers, loaders and blasters.

3 The main underground sources of dust are (a) drilling, (b) machine cutting and loading, (c) blasting, (d) loading and transferring mineral, (e) transport of mineral and rock, (f) dry picking, and concentration of mineral.

In coal mines much dust is raised by the work of coalcutters or cutter-loaders, especially those which work with a cutting chain or a percussion tool, also from shaking conveyors and by the loading of coal from chutes, or a conveyor.

The total quantity of airborne dust in a metal ore mine originates as follows from drilling up to 85%; from blasting 10%; from other work 5%.

The main source of dust formation in metal ore mines is drilling, in coal mines the work of cutter-loaders Drilling dust can be divided into three size classes (1) coarse dust, up to 1 mm across; (2) medium dust from 1 to 0 05 mm (50 microns), (3) fine dust from 0.05 mm downwards.

Fine dust, in particular that of sizes below 10 microns, mainly from 5 to 0 4 micron, is formed in large quantities during drilling, and its concentration at the face can reach 100,000 particles per cu cm of mine air in dry drilling, corresponding to some tenths of a gram per cu m of air.

To give an impression of the dust quantity formed underground at the face in various tasks we quote some numerical values obtained mainly from Soviet metal ore mines where generally speaking there is less dust than in coal mines

(a) In percussive, dry drilling of non-ferrous ore, at a single face with some 10 to 20 shotholes 1.8 to 2.3 m long, about 70 to 130 kg of dust is formed, distributed as shown in Table 2-3; (b) In the sto-

TABLE 2-3 **Quantity of Dust in the Air in a Stope after Drilling a Round of Dry Holes**

Number of dust particles per cu cm of air sample from the face	Proportions of each size, per cent						
	Up to 2 microns	4 microns	6 microns	10 microns	24 microns	50 microns	100 microns
32,600	52 6	24 1	10 2	8.9	3 0	0 25	0.2
34,800	85 0	10 0	2 0	1 6	1 0	—	—
108,250	86 1	10 0	2 5	0 7	0 2	—	—
120,780	82 4	12 5	4 2	—	0.7	—	—

pes in thin veins, drilling from 1.1 to 3.5 m of shothole per cu m of ore raises from 3-4 to 11-12 kg of dust per cu m of ore, (c) The quantity of dust formed in percussive drilling depends mainly on the hardness of the rock, its wetness, and the air pressure; (d) The harder and dryer the rock, other things being equal, the finer will be the dust; in dry drilling of hard rock, 1 cu cm of air at the face contains up to 50,000 suspended particles smaller than 1 micron; in medium hard rock 10,000 to 15,000, in soft rock 1000 to 2000, (e) The quantity of suspended dust after blasting reaches (according to the observations by Kovshul and Nedin at the metal ore mines of Krivoi Rog) 100,000 to 200,000 particles per cu cm after 15 minutes; after 60 minutes the dust count falls to 15,000-30,000; (f) During the loading of dry ore into mine cars, airborne dust concentrations have reached 60 to 340 mg/m^3, (g) The quantity of suspended coal dust underground during such tasks as machine coalcutting, shaker-conveying, transfer of coal from conveyor to mine car and in particular the work of cutter-loaders is expressed not in milligrams but in grams per cu m of air, reaching 5 to 10 g/m^3, sometimes even more if no dust suppression is practised

In the USSR, mine air is tested for its dust content by the weight method with the aid of aspiration ejector devices Types АЭР-4, АЭР-4М, АЭРА. The operation of these devices is based on the sucking of air through a dust tube with filters made of cotton or other materials, using an ejector supplied with compressed air The АЭРА device is provided with automatic regulation of air flow.

In the АЭР-4 and АЭР-4М, the volumetric speed of air is maintained between 10 and 30 litres per minute.

Dust samples are taken with special type АФА-В-18 filters made of hydrophobic fabric ФПП and placed in a special metal or plastic sleeve. Sometimes, use is made of cotton filters placed in glass tubes. The treatment of cotton filters takes much time. Dust samples taken with aspiration devices are processed in a laboratory

The quantity of air passed through the tube is obtained from the formula

$$Q = v \cdot t \tag{2-1}$$

in which Q = volume of air drawn through the tube, litres
v = flow rate of air passing through the flow meter, litres per min
t = duration of sampling, minutes.

If n mg of dust is filtered out of the air volume Q the dust content in the mine air will be

$$N = \frac{n}{Q} \text{ mg/litre} \tag{2-2}$$

or

$$N = \frac{n \cdot 1000}{Q} \text{ mg/m}^3$$

Example A dried tube containing cotton and metal mesh for sampling weighed 25 g After sampling and a second drying, the tube weighed 25.0625 g. The dust in the tube therefore amounted to 25.0625 — 25.0 g — 0.0625 g.

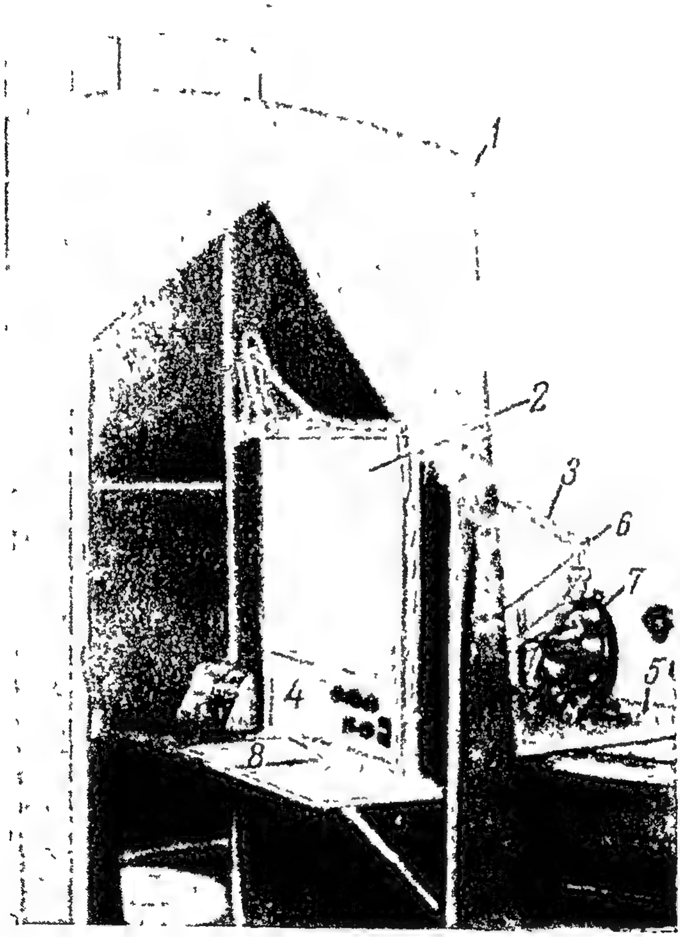

Fig 2-1 The NIGRIzoloto Institute projection apparatus

1—cabin, 2—screen, 3—frame of screen, 4—panel for remote control of microscope, 5— base of screen frame, 6—projector from microscope, 7—microscope, 8—eight-column ruled paper

Air was drawn through the tube at the rate of 22 litres per minute for 30 minutes The volume of air drawn through therefore was

$$Q = 22 \cdot 30 = 660 \text{ litres}$$

The dust content in the mine air

$$N = \frac{62.5 \cdot 1000}{660} = 94.7 \text{ mg/m}^3$$

The particle distribution of dust samples taken with ФПП fabric filters is determined under a microscope after subjecting them to special treatment. In the microprojector designed for this purpose (Fig. 2-1) the microscopic image is projected onto a screen.

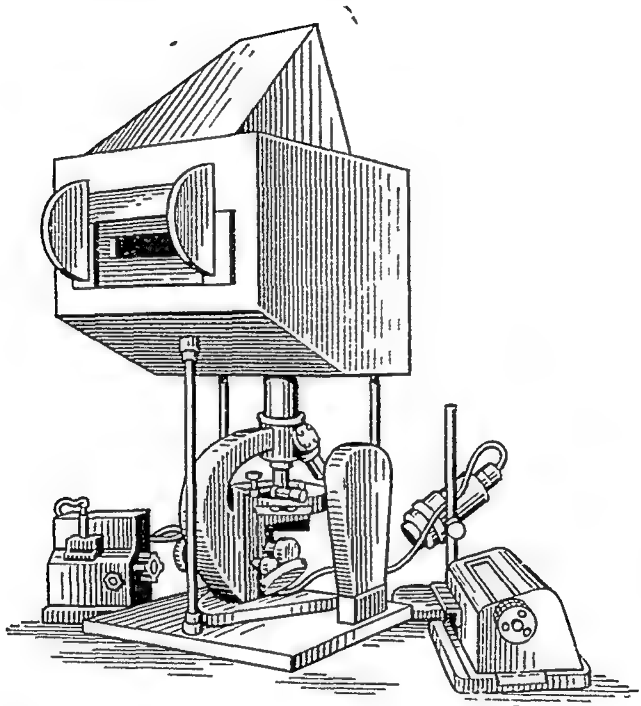

Fig 2-2 Portable projection apparatus for microscope work, developed by the Mining Institute of the USSR Academy of Sciences

The apparatus enlarges the image of the dust 3.2 times more than the usual microscope used for this purpose and also increases the number of visible particles. The object glass has a magnification from 8 to 90 and the eyepiece has sevenfold magnification.

The portable projection chamber developed by the Mining Institute of the USSR Academy of Sciences (Fig. 2-2) has a fitting for moving the sample a predetermined distance, enabling the number of dust particles to be easily and simply determined by both continuous and selective counting.

To obtain the pollution factor from a sampler sheet, use is made of rapid-action photoelectric dust counters such as Ф-1, ДПВ.

These instruments operate on the principle of measuring the attenuation of the light flux passing through the dusty air or a dust preparation Photoelectric instruments are much inferior to aspirational ones as regards their precision.

2-3. THE CONTINUOUS ULTRAMICROPHOTOMETER

Considerable attention has recently been paid to the development of methods of mine dust estimation which do not require the separation of solids from the air. One of these instruments is the ВДК continuous ultramicrophotometer designed by the Soviet scientists B V. Deryagin and G.Ya. Vlasenko The light flashes occurring

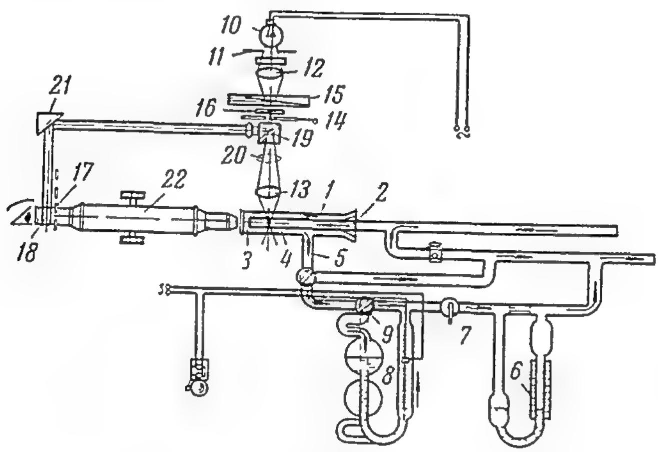

Fig. 2-3 Layout of the ВДК continuous ultramicrophotometer

1—glass vessel, 2—duct, 3—brightly lit zone, 4—outer ring-shaped space, 5—tube, 6—flow meter, 7—fine-adjustment valve, 8—integrating flow meter, 9—three-way cock, 10—car lamp, 11—heat filter, 12 and 13—lenses, 14—adjustable slit, 15—optical wedge, 16—counterwedge, 17—comparison star, 18—glass plate, 19—semi-transparent beam splitter, 20—lens, 21—reflecting prism, 22—microscope

during the flight of the dust in the air sample are counted under the microscope while the dust is drawn through a brightly lit zone in a special glass vessel. If the number of flashes is divided by the volume of the air sample, the dust count is obtained.

The instrument makes it possible to determine both the particle concentration and the particle size distribution of the suspended dust The latter is done by multiple counting of the flashes with known, stepped, diminishing degrees of illumination in the lighted area, obtained by interposing a grey photometric wedge in the path of the light rays The ВДК instrument is shown diagrammatically in Fig. 2-3.

The air sample flows through the tube to the glass container and its internal duct, passing through the brightly lit zone and along the outer ring-shaped space through the tube *5* into the monitoring part of the instrument The flow rate of the air through the glass container is determined by observing the levels in a calibrated flow meter and is varied by a fine-adjustment valve. The total volume of the sample passing through the glass container during the counting of the flashes is determined by an integrating flow meter connected by a three-way cock.

The brightly lit zone in the central part of the glass container is provided through a specially built illuminator, from a low-voltage automobile lamp The illuminator contains a glass heat filter *11*, a system of lenses *12* and *13*, an adjustable optical slit *14*, the movable optical wedge *15* and its counterwedge *16*.

To eliminate the errors which could result from variations either in the light source or in the sensitivity of the observer's eye, a standard of light dispersion is adopted—a "comparison star' located in the focal plane of the eyepiece and illuminated by the same light source.

This "star' is a small flaw on the plane surface of a glass *18*, illuminated by a pencil of light from the semi-transparent beam-splitter *19*, through a lens *20* and a reflecting prism *21*. The flashes from the airborne dust particles passing through the illuminated zone in the glass container are counted by the observer through the microscope *22*

The ВДК instrument can measure concentrations corresponding, in the weight method, to a range from fractions of 1 mg/m^3 to hundreds of mg/m^3. It can detect particles between 10^{-6} and 10^{-3} cm in diameter.

Neither the instruments described above, nor any others now in use for measuring the dust in mine air, give a true dust content. They are all to some degree selective, some of them catching only the coarse fraction of harmful dust (filters, konimeters), others catching fine dust Their great disadvantage also is that almost all of them (except the ВДК continuous ultramicrophotometer and the MakNII photoimpinger) are unsuitable for direct dust determinations in the air; they are only suitable for taking samples which must be processed later. Recent work on the design of optical (Tyndall effect) and electronic instruments for directly estimating the dust particles in the air is therefore of special interest.

2-4. METHODS OF REDUCING THE DUST HAZARD

Dust suppression underground is difficult and complicated, requiring a number of technical, medical and social measures.

The technical measures are divided by their function into two groups.

1 Measures aimed at reducing the formation of fine dust. These include hydraulic mining, methods of mining with heavy blasts or with long-hole blasting in caving, and methods of dustless drilling by thermal, high-frequency, or vibration methods, etc

2. Measures reducing the dust content in the air underground, or the suppression of airborne dust.

The first group of measures, notwithstanding their effectiveness, cannot reduce the dust formation down to permitted levels underground and therefore dust suppression in modern mines is mainly effected through the second group of measures with the following main methods: (a) effective ventilation of the workings, particularly the faces, and places near them; (b) wet drilling, with special wetting agents in the drilling water, (c) dry dust catching when there is difficulty with wet drilling; (d) water spraying in dusty places by means of hoses, water curtains, etc.; (e) preliminary water infusion into the coal under pressure; this reduces dust formation and the amount of airborne dust.

Dust suppression measures are aimed at: (1) ensuring the supply of fresh air to the faces in amounts required in accordance with the dustiness coefficient; (2) preventing the blowing of dust off the walls and floor of the workings, the blasted and hauled minerals or rocks, (3) ensuring, along with other dust-fighting measures (wetting of the coal face, water spraying), the maintenance of the appropriate atmosphere at the faces in accordance with the permissible sanitary rates of dust concentration

Efficient ventilation is attained principally by the joint effect of two factors, namely the liquefaction of the dust cloud by turbulent diffusion and the removal of airborne dust by the air flow

In practice, three distinct kinds of dust conditions may be encountered: (1) When the intensity of dust concentration in the atmosphere does not increase with increasing velocity of air flow, or when it increases to a smaller degree than the intensity of air supply. In this case, the dust concentration falls off due to the liquefaction (2) When the velocity of air flow is such that the intensity of dust concentration compensates for the intensity of liquefaction, while the dust concentration remains constant. (3) When an increase in the velocity of air flow causes larger and larger dust particles to become airborne, and it begins to be blown off the floor and walls of the workings, the blasted and hauled minerals. The dust concentration in the air rises.

The effectiveness of the dust suppression action of the ventilation depends on the above-described dust conditions.

As a result of numerous investigations, the following limiting optimum velocities of air flow have been established.

> at faces and in chambers—1.4 to 1 8 m/sec;
> at developments—not less than 0 4 to 0 5 and not more than 0 7 m/sec.

When technical measures for dust suppression are used effectively, these limits may be somewhat extended in each specific case.

In hauling and conveyor crosscuts, in braking and other inclines equipped with conveyors, the optimum velocity of air flow is determined with an allowance for the conveyor speed, on the basis of the rated flow in accordance with the dust factor.

The airing of dead-end workings is best achieved by forced draught, because in this case the fresh air is not polluted with dust on its way to the face.

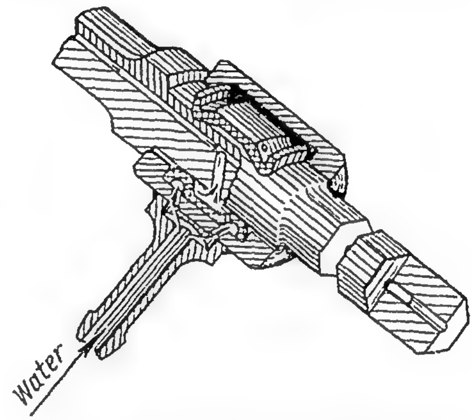

Fig. 2-4 Coupling for feeding water into the side of a drill steel

For wet drilling to be effective the following conditions are necessary (a) the water must be clean, without solids or bacterial impurities; (b) there must be enough water, it is desirable to have wetting agents added to it, substances which lower the surface tension of the water and make it flow more easily along the surface of the dust particles; (c) the construction of rock drills and pneumatic picks must be such that air cannot enter the hole with the water because this results in the formation of an emulsion which diminishes the efficiency of dust wetting. This emulsion is much less likely to be formed if the water is fed into the side of the drill steel, avoiding the rock drill itself; the water flows directly down the central hole in the drill steel from a special sleeve on the steel (Fig. 2-4).

With hand-held rock drills, the water flow should be about 3.5 to 5 litres per minute and for column-mounted rock drills about 5 to 7 litres per minute. The wetting agents used are (in concentrations from 0.1 to 1.0 per cent by volume to 100 volumes of water): naphthenate soap, manufactured at petroleum refineries, ДБ composition which is a non-ionic, surface-active substance, an oily liquid, readily soluble even in hard water; ОП composition, a compound of polyethylene glycol monophenyl alkyl ether.

The solution of the wetting agent is made on the surface and then brought underground to the working place. None of these wetting

agents can be used everywhere in all drilling conditions with dusts of any petrographic composition. The most effective agent is ДБ, judging by the careful tests made in the laboratories of the Institute of Physical Chemistry of the USSR Academy of Sciences.

The use of water for dust suppression is generally exceedingly effective, specially for wet drilling, it can lower the airborne dust content down to the permitted level. However, under certain conditions the use of water is difficult or impossible, for instance in remote faces with no water main, when the rock being drilled would swell or absorb water, in the upward drilling of holes, when there is little headroom and holes must be drilled by men lying down; when drilling in permafrost or in hot deep mines where the air humidity must not be increased

In these cases the best method is to drill dry and to trap the dust. A large number of dust traps have been developed, generally on the same operating principle sucking out the dust from the drill hole with devices additional to the rock drill which may be of the ordinary type or of special construction, the dust being sucked out through a hosepipe to a dust-settling and filtering device which is periodically cleaned out, or sometimes to a waste area. Some of the dry dust traps are extremely efficient. For example the СПН-5 dust catcher (NIGRIzoloto, designed by engineer Sipyagin) trapped up to 99.9 per cent of the dust in laboratory tests.

The usual dry dust trap is designed to serve one rock drill. In recent years a method of centralized dust trapping in dry drilling has been developed and tested in the USSR. This method has two variants:

1. Trapping of the dust by one central stationary trap;
2. Trapping of the dust using old workings.

For the first variant of the centralized dust trapping method, special stationary traps have been developed including the СПН-7 dry dust catcher (by NIGRIzoloto) and the ВНИИ-1 dry dust catcher (Magadan) and others

With the second variant the dust formed when drilling the holes is sucked out of them through a hole in the drill steel and then through the dust exhaust hose into a pipeline. The dust pipeline is operated at the flow rate of 20 to 25 m/sec, the air passes to the dust settlement area where its flow rate drops to 0.02-0 03 m/sec. Because of this abrupt drop in the air flow rate a large part of the dust particles separates out of the air.

For centralized dust extraction from a large number of rock drills the ЦСПУ-2 dust catcher is used in the USSR (Fig. 2-5). It cleans the air in four stages, the first stage being a cyclone, the second an outlet filter with two meshes, in the third it passes through a water layer, in the fourth through a spray filter made of

porous sponge rubber and located at the air outlet from the tank to the atmosphere. The residual dust in the air does not exceed 0 1 to 0.2 mg/m³. The liquid slurry flows to the mine drainage system. The air is drawn in by an electrically driven vacuum pump type PMK and it can transport the dust for 200 m or more. With the PMK-2 pump, the dust catcher can serve 5 to 10 rock drills, or up to 30 or more drills with the PMK-4 pump.

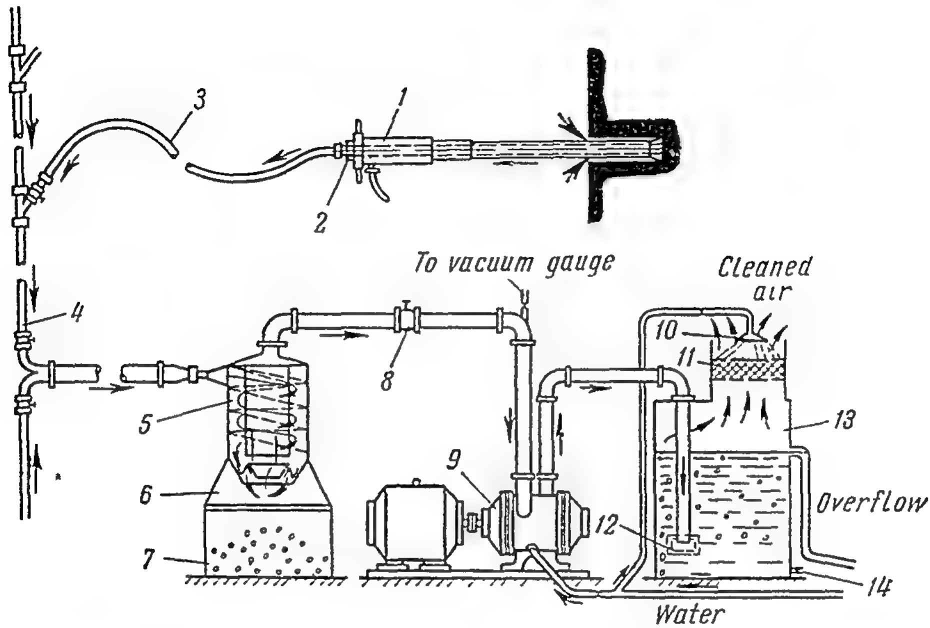

Fig 2-5. Diagram of the ЦСПУ-2 dust-catching plant
1—drill with dust extracted centrally, 2—dust-connection fitting, 3—hose for removal of dust, 4—vacuum pipeline, 5—cyclone, 6—shut-off valve, 7—dust bunker, 8—valve for regulating the vacuum in the system, 9—vacuum pump; 10—sprayer, 11—filter, 12—screen, 13—tank, 14—drain cock

Electrostatic precipitators and electric neutralization of dust are also of practical interest. In electrostatic precipitators, the dusty air passes through a high-tension electric field. The dust particles become negatively charged and are attracted to the positive electrode where they settle, and give up their electric charge. Direct current at 75,000 volts is used, the efficiency of electrostatic precipitators reaches 99 7 per cent if the air is dry.

Dust removal by electric neutralization is based on the fact that particles of different substances when airborne for a short time in a turbulent flow acquire electric charges of different magnitude and sign.

When mine dust of one composition is acted on by a dust of different composition which receives a charge of opposite sign during

crushing, it is possible to coagulate the particles and so increase their weight. Losing their ability for thermal movement, they separate out of the mine air It must be remembered that the smaller the dust particles the greater their mobility and activity and the more easily they are neutralized.

It has been proved theoretically and in practice that dust can be precipitated from the air by high-frequency sounds (ultrasonic precipitation).

To precipitate very fine and particularly hydrophobic dusts, a new condensation method is of interest, based on the condensation of water vapour on fine solid particles, making them heavier and thus precipitating them from the air.

Condensation of water vapour on the dust can be attained by several methods mixing the airborne dust with live steam in a turbulent flow*; mixing damp dusty air with a jet of cold air; cooling the surface of the pipe along which the damp dusty air is moving, etc.

The efficiency of the condensation method was proved in laboratory experiments at the NIGRI Institute (Krivoi Rog) in a small condensation chamber using dusts from quartzite, shale and coal, and was extremely high; the air introduced into the chamber with 20,000 to 40,000 dust particles per cu cm was almost completely cleaned in 10 to 15 minutes provided that the air in the chamber was kept supersaturated with moisture.

Further experiments are however needed to assess the practical value of the condensation method, mainly to clarify the possibility of cleaning large volumes of air and the economic expediency of this method.

To settle the airborne dust from shotfiring it is highly advisable to carefully water the face and the area near it in combination with a water spray curtain at the return side of the working place. Observations in various metal mines have shown that this method has lowered the airborne dust content at the face by 70 per cent or more. Spraying is also effective during the loading and transport of ore and coal.

As was pointed out above, a large amount of dust is formed in coal mines during the operation of coalcutters and particularly of cutter-loaders. Coal dust, though one of the least harmful of dusts, not only hinders work at high concentrations but is also unhealthy, causing anthracosis. Water spraying is therefore used in coal mines for suppression of airborne dust during the work of coalcutters and cutter-loaders.

---

* Experiments on dust suppression by steam have shown the efficiency of this method and a number of its disadvantages, such as the need for a portable electric boiler, the high power consumption of 20 kilowatts per rock drill, etc.

Fig. 2-6 shows the water-spraying system of a coalcutter, with two pipes *1* placed in special ducts within the jib of the machine and connected on one side to two water jets *2*, and on the other side to the water supply pipe *3*, fixed to the body of the machine and supplied with water from a hosepipe in the face. Observations in

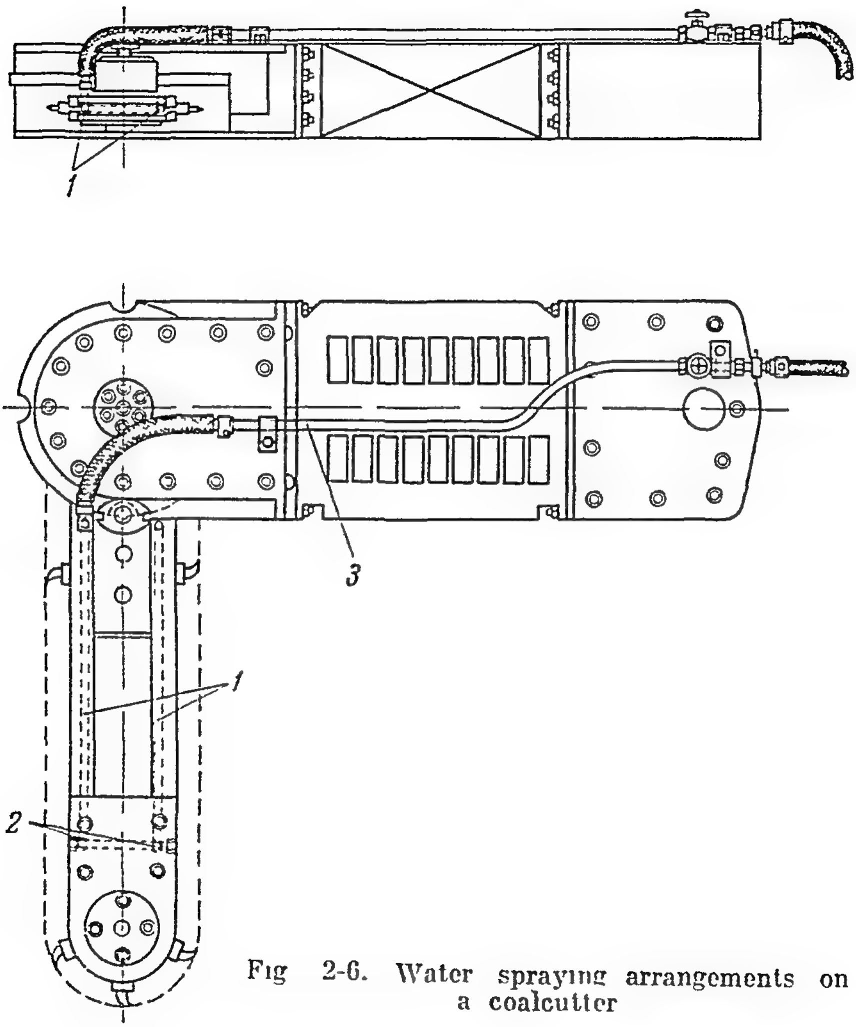

Fig 2-6. Water spraying arrangements on a coalcutter

the Donets Basin coalfield at water pressures of 3 to 5 atmospheres and using 16 to 18 litres per metre length of cut showed reductions in the airborne dust at the face from several grams per cubic metre down to 50-100 mg/m³ of air.

The spray device of the Donbass cutter-loader is designed as follows (Fig. 2-7): on the rear side of the rotary loader is a filter with a cock (not shown) to which water is supplied from a pump; from

the filter through the tee-piece *1*, water is brought through flexible rubberized hoses to the three jets *2*, mounted under the top plate of the loader jib Simultaneously through the hose *3*, the water reaches the jets *4* mounted at the drive sprocket of the reduction gear

The efficiency of watering increases if the water contains one of the wetting agents which were discussed on page 115.

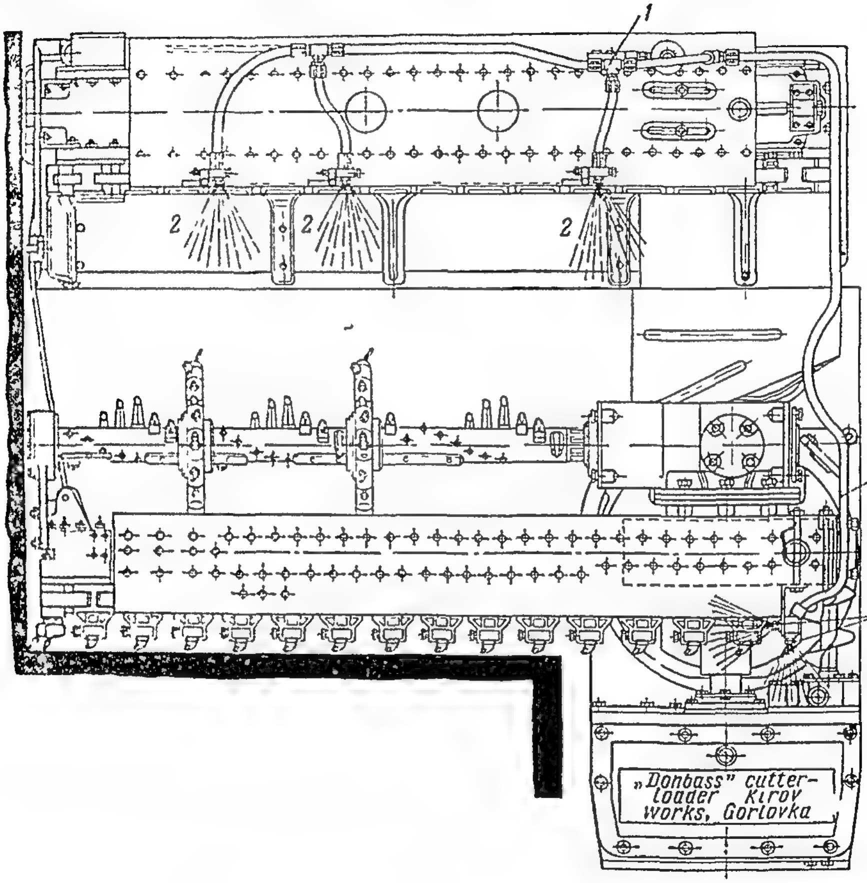

Fig 2-7 Water spraying on the "Donbass" cutter-loader

Recently both in the USSR and in other countries a method of dust suppression has been widely used, involving the injection of water into coal seams before the coal is broken, which when properly used reduces the airborne dust by 70-80 per cent Water infusion, as the method is called, is obligatory under the Safety Regulations; this rule is waived in seams where the permitted dust level is maintained by other methods of dust suppression as well as in seams

which are ordinarily damp and do not need water infusion, and finally in seams where the containing rocks do not allow the use of water.

Two methods of water infusion into the coal mass are used (a) through short boreholes, and (b) through long boreholes

When the water is injected through short boreholes these are generally drilled from the coal face; when long boreholes are used they are drilled either from the coal face or from the return airway, parallel to the face line.

The short holes are usually drilled perpendicularly to the face line and if the seam is homogeneous they are driven in the middle of the seam, if the seam has bands of different hardness the holes are drilled in the hardest band. The distance between holes depends on the water permeability of the seam and is determined experimentally. Usually it is from 1 5 to 3.0 m, rarely more, the holes being usually 2 m deep, sometimes longer. The infusion pressure depends on the water absorption of the seam and is:

in highly permeable (soft) coals . .	up to 15 atm
in permeable (medium hard) coals	15-50 atm
in seams of medium permeability (hard) coals	50-120 atm
in impermeable (very hard) coals	120-200 atm

The water pressure must be such that it ensures the maximum saturation of the coal mass with water in the shortest time for each hole (10 to 20 minutes) without breaking out the coal from the mass before it is saturated with water. The time of infusion varies widely, it is usually not more than 30 minutes but sometimes reaches three hours or even more The rates of infusion are usually about 1 litre per minute (Donets Basin), in other basins 5 to 15 litres per minute or even more. The water consumption per hole in the Donets Basin is usually up to 50 litres, occasionally up to 100 litres and very rarely up to 250 litres. In other countries 250 to 300 litres or more are used

Water infusion layouts vary widely. For example from the district water main at the haulage road, the water passes through a filter to the water main along the face from which, through tee-pieces with a valve and a pressure hose, it passes into a portable pump driven by the motor of a hand-held electric drill and then to the high pressure hose at the borehole. Other arrangements are possible.

One sign of the end of infusion is the appearance of water in the face or in neighbouring boreholes. Usually before establishing the infusion parameters (the distance between boreholes, the water pressure and so on) some tests must be made. In water infusion

through long boreholes drilled from the coal face the following infusion parameters are recommended·

borehole diameter 45 mm,
spacing of boreholes 10-15 m,
depth of boreholes 10-15 m

The advantages of this method are that the water penetrates the coal mass uniformly, saturates it with little labour and economically.

Before the water is injected the collar of the hole must be sealed and various arrangements are used for this purpose. To wet the coal more easily, some 1 to 1.5 per cent of wetting agent should be added to the infusion water

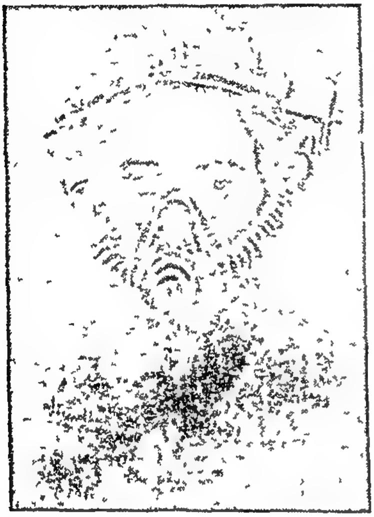

Fig 2-8 The PH-21 respirator

When the water is injected through long holes drilled from the return airway they are of 10 mm diameter, spaced usually at 10 to 15 m, and they are long enough to reach some 15 to 20 metres from the haulage road. The depth of the seal is 7 to 8 metres, the water consumption being not less than 20 to 30 litres per ton of coal and the pressure not less than 150 atmospheres.

All these measures for suppressing airborne dust in the working places, when combined and conforming to the standard, are highly effective, but under some working conditions combined dust suppression is difficult Individual dust protection is then used and miners working in dangerous dust concentrations wear respirators to filter the dust from the air. There are about ten types of these respirators in the USSR.

An efficient and comfortable respirator is the PH-21 of the NIGRIzoloto Institute (Fig 2-8) The air is cleaned in it by four filters, two on each side of the face, made of special ФМП-Н board. Each pair of filters is set in a plastic body having four openings for the passage of air.

The resistance of the respirator to breathing, with clean filters, is not more than 3.8 mm H_2O and at the end of the shift not more than 4.8 mm H_2O. The dead space of the respirator is 65 cu cm.

In the ventilation laboratory of the Mining Institute of the USSR Academy of Sciences a similar respirator has been developed but with a filter containing charcoal grains*.

* The recently developed anti-dust respirator, type ПРШ-1, should also be mentioned, it was designed by the Central Scientific Research Laboratory

At the surface of the mine a large amount of dust is formed on the picking belts and near the fine crushers, etc. To prevent the transfer of dust, such items of equipment, including tipplers, are covered with hoods and the dust is sucked out from them by exhaust fans sending it to dust collectors; water spraying is also used to settle the dust if this does not make subsequent processes of picking and cleaning more difficult and does not deteriorate the product. An important requirement in Soviet regulations is that the intake air to the mine must not pass through the sorting plant nor near the tipplers, and moreover rock tips and tailings of preparation works must be placed at an adequate distance from downcast shafts.

Apart from these purely technical dust suppression measures to protect miners from breathing in dust, particularly siliceous dust, medical and social action is needed to maintain the miners' physical resistance, and to diagnose any dust disease, particularly silicosis as early as possible, by periodical medical inspections of all miners who work in dusty places, using radiography and the necessary laboratory investigations, improved supervision of the health conditions at work; high-caloric diet, holidays or transfer to work on the surface of the mine.

These preventive measures are obligatory under Soviet legislation for all work where the rock or mineral contains more than 10 per cent of free silica but they are needed to some degree for all dusty mining work, independently of the dust composition.

Overall supervision of silicosis prevention among miners is carried out by the Soviet Mining Inspectorate and the Ministry of Public Health. A special committee of the Soviet Academy of Sciences on silicosis* supervises and coordinates the investigation of this problem.

2-5. EXPLOSIBILITY OF MINE DUST

Explosibility of various dusts. Very many solids which ordinarily are incombustible or not considered easily flammable, become highly flammable or even explosive when they are reduced to the

* To conclude this review of measures to combat dust as an occupational hazard we must mention also the prevention of silicosis by inhaling aluminium dust

In the post-war years information was obtained on some Canadian metal mines in which aluminium dust was successfully inhaled to reduce the incidence of silicosis; in some Soviet metal mines the same method was introduced, using the miners' changing rooms where the men spend about ten minutes before going underground At the four corners of the room before the men come in, compressed-air injectors spray aluminium powder into the air at the rate of 1 gram per 28 cu m of room

A similar inhaling method, not with aluminium powder, but with common salt was tried in some West German coal mines, with positive results

state of fine dust. This applies to some metals aluminium, iron, zinc, alloys such as bronze, pyrite ores and many other solids such as flour, starch, sugar, malt, cocoa, cork, wood, coal, etc.

This is explained as follows (a) the surface area of the dust in contact with oxygen is enormous*, and the dust particles can thus absorb tremendous quantities of oxygen, (b) combustible gases are liberated from some dusts when heated

A cloud of fine dust, heated at any point up to its ignition temperature, rapidly ignites throughout its volume An explosive burning takes place with increasing pressure and speed of propagation of the flame. This burning can change into a real explosion. Particularly dangerous are those dusts which (a) release combustible gases, for example bronze powders (used in decorating) which decompose the moisture of the air, releasing more than 1100 litres of hydrogen per kg of powder, or coal dusts which on heating release 200 to 300 litres of various combustible gases per kg, (b) can oxidize intensively at relatively low temperatures (e g dust of various charcoals from cork, wood, etc ; pyrophorous dust which at temperatures above 100-150°C rapidly heats and takes flame).

It must also be remembered that a cloud of dust becomes charged with static electricity because of the friction between the dust particles and because the finest particles are in a state of constant motion, and in favourable conditions it can become discharged to the same extent as a thundercloud. The discharge of this cloud accompanied by sparks and flame can start a spontaneous explosion, without any external igniter

The probability of dust electrical discharges underground is low but in factories where various substances (particularly organic) are finely ground, such as flour mills, or factories for powdered sugar, cocoa, cellulose, artificial rubber, or briquetting works, etc., the

* The increase in surface area of solids when crushed can be seen from the table below If a piece of a solid, for example coal, is cubic in shape with edges 1 cm long, and the volume of 1 cu cm is transformed by crushing into cubes of side 1 mm long and subsequently 0 1 mm long and so on, the total area of the cubes obtained by the crushing is as follows

Length of side of cube	Number of cubes	Total surface area of cubes, sq cm
1 cm = 10 mm	1	6
1 mm	10^3	60
0 1 mm	10^6	600
0 001 mm = 1 micron	10^{12}	60,000

formation of "thunderclouds" and spontaneous explosions in a hot dry atmosphere is possible and has taken place.

A dust cloud can be very easily charged with static electricity if the air containing the dust moves at some speed along a pipeline. This was established as long ago as 1927-1931 in Britain by investigations into the movement of dusty air along mine ventilation ducting, also with compressed air moving along ordinary pipelines. The experimental apparatus consisted of steel pipes (dia 129 mm and length 5.5 m) set horizontally at 1.5 m from ground level on two ebonite bearings fixed on to glass bases. The air inlet end of the pipe was joined, by means of an ebonite pipe 253 mm long and 127 mm in diameter, with a small fan provided with a device to enable the air speed in the pipe to be varied from 2 to 14 m/sec. The dust was automatically injected into the air from a small container near the fan intake The electrical capacity of this system was about 0 0002 microfarad The experiments were made with very fine pure coal dust from various seams, also with fine Fuller's earth and with mixtures of coal dust and fine clay. The results are given below.

1. When the dusty air travelled along the insulated tube even in a fairly small quantity (for example 1 1 to 4 5 g dust per cu m of air) and at speeds from 2-3 up to 13 m/sec the pipe became charged within a few seconds up to 6,000-7,000 volts or more, producing sparks which easily ignited an 8 5 per cent methane-air mixture.

2 No electrical charge was observed when there was the slightest disturbance of the insulation of the system from earth, or at any relative humidity of the air above 65 per cent even if the insulation was perfect.

The experiments on insulated metal tubes carrying compressed air (at 3 to 7 kg/cm^2 pressure) with additions of fine sand showed that the dusty air became highly charged and sparks issued from the pipe. When an insulated screen, for example a copper plate, was placed at the outlet end of the pipe, the plate rapidly acquired such an electrical charge that sparks issued from it which invariably ignited a methane-air mixture An electrical charge was noticed even without artificial additions of dust to the compressed air but the charge was much smaller and was evidently obtained from the dust already in the compressed air.

This shows that a considerable self-induced charge of the dust in dusty air is possible only with very dry air and with perfect insulation to the pipe containing the dusty air and to the object which is struck by the air flow. These conditions, if they are ever possible underground, occur very rarely indeed Consequently there is no need to overestimate the practical danger from self-induced charges in airborne dust; nevertheless this property of dust must be remembered, and all pipelines, machines, etc. must be earthed.

2-6. COMBUSTIBILITY AND EXPLOSIBILITY OF COAL DUST

In 1803, over 150 years ago, it was proved by the Wallsend explosion near Newcastle, England, that coal dust can take part in mine explosions. The flame of the explosion was evidently transmitted to another district of the mine with the participation of the coal dust. Notwithstanding the investigations then begun and continued later into the explosibility of mine dust, the opinions of mining specialists, whether theorists, or practical men, diverged, particularly on whether coal dust can take part in underground explosions in the absence of gas.

The coal dust explosion in 1906 at the non-gassy mine of Courrières in northern France, sacrificing 1100 lives, ended the argument between the opponents and the supporters of the independent explosibility of coal dust, and the subsequent wide investigations into this subject finally established that

(1) dust can explode in the complete absence of methane;

(2) dust can transform a small methane explosion into a vast explosion;

(3) the flame of an ignited dust cloud can pass to a gas accumulation and ignite it,

(4) the presence in the air of fine dry coal dust lowers the lower limit of explosibility of the methane-air mixture to well below the ordinary 5 per cent;

(5) when coal dust takes part in an explosion, its combustion products always contain a considerable quantity of carbon monoxide, and experience of underground explosions has shown that most of the men killed in them (70-80 per cent) are poisoned by carbon monoxide.

The combustibility and explosibility of coal dust resemble these properties in methane

1. For dust, as for gas, there are lower and upper limits of explosibility in air

For gas, CH_4	For coal dust
5 to 6 and 14 to 16% by volume	30 to 40 g/m³ of airborne dust *
from 35-40 to 95-110 g/m³ of air by weight	up to 1500-2000 g/m³ or even more, of settled dust

2. The quantity of heat emitted during explosions of coal dust and of methane and the temperature of explosion.

* According to recent experiments by MakNII it has been established that a cloud of dry coal dust with a high content of volatile and bituminous material can explode even at 5 to 10 g/m³

1 kg CH_4, burning to CO_2, releases 13,300 kcal of heat. Coal dust, if it is regarded as amorphous carbon, releases:

(a) burning to CO_2: $C+O_2=CO_2+8,140$ kcal per kg carbon

$$12\ g+32\ g=44\ g$$

(b) burning to CO: $2C+O_2=2CO+2,440$ kcal per kg carbon

$$24\ g+32\ g=56\ g$$

The greatest quantity of carbon which, according to theory, can burn to CO_2 in 1 m^3 of air is given below: taking the weight of 1 m^3 of dry air to be 1,293 g at 760 mm pressure and 0°C, and its oxygen content to be 23 per cent by weight:

$$1,293\times\frac{23}{100}\times\frac{12}{32}\approx 112\ g$$

The quantity of heat emitted is about 900 kcal and the temperature at the point of burning reaches 2300 to 2500°C at constant volume.

In the burning of 112 g of carbon to CO about 550 kcal of heat are released and the explosion temperature is from 1300 to 1700°C.

Coal dust contains, in addition to carbon, various gases and impurities, and when it burns the actual heat effect is different but the analogy with methane is generally maintained.

3. The ignition temperature of methane is 650 to 750°C, that of fine dry coal dust is 700 to 800°C, and that of some organic dusts is even lower, 500°C. It follows therefore that all hot sources which ignite methane can also ignite a cloud of coal dust.

In the past it was frequently disputed whether such an insignificant source of heat as a miner's lamp could ignite coal dust but after the explosion at Courrières, underground dust ignitions were carefully reported and investigated, and the possibility was established and confirmed by tests mentioned in the literature.

1. In the coal mine near Rockspring in Wyoming, USA, a rope broke on an incline, and a train of wagons full of coal ran away down the incline. The wagons struck the wall of the working, and raised a thick cloud of dust which moved into a place where some naked-light oil lamps were hanging; in the subsequent explosion, 40 miners nearby were burnt.

2. In Britain in the Middleton mine in the Beeston seam, a foreman, who was in the intake airway at the underground lamproom near the beginning of the haulage road, opened his oil safety lamp so as to clean the soot from the wick. When he had finished this task, a train of coal, passing beside the door of the lamproom where he stood, raised a large cloud of dust. The last pieces of burning soot from the wick fell into the cloud of dust and ignited it. No

explosion took place but a flame about 80 cm high and 100 cm wide followed the train until it stopped about 15 metres further and the flame died out

Apart from its similarities with methane, coal dust differs appreciably from it in the following respects

1 Methane, when it is released from the coal or rock underground, always remains mixed with the air. The quantity of dust suspended in the air in even the dustiest workings is generally only a fraction of that content at which the dust cloud becomes explosive. The bulk of the dust lies on the floor and walls of the workings and to form an explosive cloud it must be raised into the air.

2 Gas is easily detected at a content several times as low as the lower limit of explosibility in air, but dust which is in a dangerous quantity in the mine is often difficult to notice. We saw above that the largest quantity of coal dust (amorphous carbon) which can burn to carbon dioxide is theoretically 112 g/m^3.

Let us consider a roadway of 2 × 2 metres cross section containing a quantity of deposited coal dust equivalent to 112 grams per cubic metre of its volume, i e., $\frac{112 \times 4}{8} = 56$ grams per square meter, and because 1 cu m of fine coal dust weighs roughly 600 kg, the thickness of the layer of coal dust round the roadway will evidently be equal to 0.093 mm, i.e , less than 1/10 of 1 mm, although the working will be charged to the limit theoretically corresponding to the maximum intensity of explosion.

3 The flammability and explosibility of methane are generally more or less uniform, at least in coal mines But the degree of flammability and explosibility of the coal dust formed underground and settled in the workings, fluctuates within very wide limits; some dusts are practically non-flammable, but others are no less flammable and explosive than methane, any property of mine dust depends on a whole series of factors such as the fineness of the dust, its content of flammable volatiles, ash content, and other factors

4. A coal dust cloud can become electrically charged spontaneously, and, in favourable conditions, can discharge, spark and thus ignite itself

5 In a methane explosion, carbon dioxide is formed almost entirely; in a coal dust explosion a large amount of carbon monoxide is almost always formed.

2-7. FLAMMABILITY AND EXPLOSIBILITY OF COAL DUST

As pointed out above, the explosibility of coal dust settled in the mine workings varies widely and is controlled by many factors. The only reliable way of finding out the hazard from the coal from

a particular seam or working is therefore to make direct tests of its explosibility.

The technique of these tests will be discussed below, here we will merely point out that they are made on a small scale with laboratory equipment, and on a large scale in experimental tunnels and mines, the essence of the tests being the same: in the instrument or tunnel, a quantity of the dust to be investigated is placed, it is then made airborne by some method and the dust cloud is subjected to the action of an igniter, e.g. a surface heated to 900 to 1000°C, or to the flame from an explosion of powder or dynamite.

If some point of the dust cloud is heated to such a temperature that it burns, the consequences may be: (a) the dust will burn only at its point of contact with the igniter; (b) the flame will propagate through the cloud at a certain speed for some distance from the igniter, but without dynamic effects, i.e. the cloud ignites in part or in full; (c) the flame spreads throughout the space occupied by the cloud, with a rapidly increasing speed and pressure (dynamic effects), i.e. the cloud explodes.

Dust with the behaviour described under (a) is considered, for the purposes of the test, as non-flammable and non-explosive. Dust which propagates in accordance with condition (b), without dynamic effects can be described as flammable but non-explosive. Finally, dust in which the flame spreads throughout the space occupied by the cloud and with dynamic effects is regarded as explosive.

The following can be used as indicators of the degree of flammability of a dust (a) the distance for which the flame will spread through the dust cloud, or the so-called length of the flame, (b) the speed of propagation of the flame, (c) the amount of non-combustible stone dust to be added to the dust being investigated, which will make it non-flammable.

The following additional indicators of the degree of explosibility of a dust, in addition to the three mentioned above can also be used (a) the pressure developed at the point of explosion; (b) the minimum density of the dust cloud in g per cu m, at which it becomes explosive, (c) the density of the dust cloud in g per cu m, at which it achieves its maximum power

Since it is not possible to be absolutely certain that dust which during testing is just flammable will not become, in mine conditions, explosive it is inexpedient and extremely difficult to establish a boundary between flammable and explosive dusts, and for this reason every flammable dust should be regarded in practice as explosive.

The most generally accepted index of explosibility of a dust at present is the amount of stone dust which must be added to it to make it non-explosive. This index can be expressed in two ways: either

directly by the weight of stone dust added, in grams per gram or kg per kg of dust being tested, or by the percentage of ash content in the mixture after the stone dust has been added, including the natural ash content of the dust being tested

This stone dust must of course be completely standardized both in fineness and in composition and this will be discussed below.

2-8. FACTORS AFFECTING THE EXPLOSIBILITY OF COAL DUST

The numerous factors which affect the explosibility of coal dust can be divided into the following six groups.

(1) the fineness of the dust,

(2) its composition, the content of combustible volatiles, ash, and moisture,

(3) the quantity of dust suspended in the air and settled in the workings,

(4) the presence of methane;

(5) the type and power of the igniter,

(6) various side effects, e g the distribution of settled dust around the working, the natural humidity of the working, etc.

1 *The fineness of the dust* It is now thought that the dust taking part in explosions includes the finest dust up to particles of 0.75 to 1 mm across, but many tests have shown that *the explosion is carried by the dust fraction which passes through a No 80 sieve**, or the so-called minus 80 fraction. This dust consists of particles smaller than 1/10 or 1/15 mm across This dust is impalpable, if it is passed between the fingers it cannot be felt, and in a thin layer it is velvety to the touch. The larger the proportion of this fraction in the dust the more dangerous it is, and vice versa.

Numerous tests of the dust of the Pittsburgh seam in the USA, containing 39 to 40 per cent of volatiles, in the dry and ash-free condition, undertaken in an experimental mine, showed that this coal dust, containing about 10 per cent of particles below 1/10 to

---

* The number of the sieve is equal to the number of holes per centimetre length of the mesh so that for example, a No 12 sieve has 12 holes per cm length of mesh

The fineness of the dust is expressed by the number of the sieve or by the number of holes, using the minus sign if the dust passes through the sieve, and the plus sign if the dust remains on the sieve, indicating at the same time what percentage remains on the sieve as a residue or passes through Thus a dust expressed as 20% + 80 could also be expressed as 80% — 80, and would consist of 20% of particles remaining on a sieve with 80 holes per cm length, and of 80% of particles which pass through this sieve The linear dimension, a, of the sieve hole depends on the number of holes per unit length and on the thickness of the wire which it is made of

1/15 mm and 90 per cent from 0 1 to 0.75 mm, having about 6 per cent of ash, ceases to be explosive when stone dust is added to it to the extent of about 1 2 kg per kg of coal dust; but when it passes completely through a No. 80 sieve (finer than 1/10 to 1/15 mm) about 3.5 kg of stone dust must be added to it, that is about 3 times as much.*

The participation in the explosion of the coarse particles which do not pass through the No. 80 sieve, in particular the grains of 0 25 to 0 75 mm, is mainly affected by the extent to which they are crushed by the force of the explosion.

On the other hand the finest dust, less than 10 microns across, apparently becomes less explosive. This is explained as follows:

(1) in very fine condition, the dust decomposes into particles of different chemical composition;

(2) this dust tends to coagulate and to form flakes,

(3) very fine dust particles rapidly oxidize and thus become less flammable

2. *Composition of the dust* The explosibility of the dust varies directly with its volatile content.

The volatiles are conventionally calculated for this purpose according to the formula (on the basis of a dry, ash-free mass):

$$V = \frac{\%\ \text{volatiles by analysis} \times 100}{100 - \%\ \text{ash} - \%\ \text{moisture}} \tag{2-3}$$

Numerous experiments both in the laboratory and in test tunnels and mines have shown that dusts with volatile contents (V) less than 10 per cent (i e. lean coals and anthracites) can be considered practically non-explosive. Dusts with 10-15 per cent volatiles are slightly explosive but when the content exceeds 15 per cent the explosibility of the dust rapidly increases. The rate of increase differs with different dusts. The tests on American coals in an experimental mine yielded a curve which climbs steeply from 15 to 20 per cent and then rises smoothly (Fig. 2-9)**.

The natural ash content and humidity lower the explosibility of the dust, but their effect is considerably smaller than that of the volatile content in the opposite direction To illustrate this, we can mention No. 2 seam of the Kizel coalfield, with a volatile content V = 35.7 per cent; in laboratory tests at the Makeevka Research Institute (MakNII) this dust was extremely explosive, while its natural ash content was 18 per cent. Fine coal dust, with a large pro-

* In the first instance the total ash content of the mixture is 57 per cent. in the second 79 per cent

** Experimental work by MakNII on the relation between the explosibility of coal dust and its volatile content has confirmed the increase of explosibility with increase of volatiles.

portion of the —80 fraction, and with a high volatile content, $V = 30$ per cent, ceases to be explosive at an ash content of 60 to 70 per cent or if water is added to it in the ratio 1 : 1.

3. *Dustiness of the roadway.* The quantity of dust settled in the working in g per cu m of volume of the working is expressed as δ g per cu m, the density of the dust cloud being expressed in g per cu m of cloud, δ_0

If the coal dust is fine (from 75 to 100 per cent passing a No. 80 sieve) and has a volatile content $V = 30$ per cent, or more,

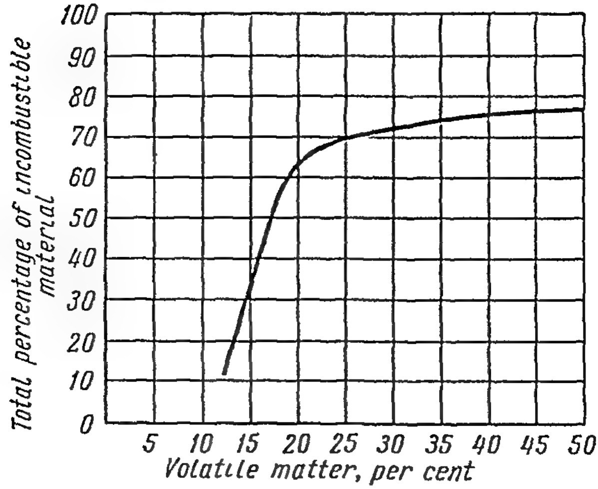

Fig 2-9 Explosibility of coal dust versus its volatile content

airborne propagation of the flame through the cloud begins at $\delta_0 = 30$ g/m^3 or even less Thus in experiments in a small tunnel at Liévin in France using fine dust, 100 per cent of which passed a No 80 sieve, from the Courrières mine with a volatile content of 30 6 per cent, the flame began to propagate when the cloud had a density $\delta_0 = 23$ g/m^3

Similar observations were made during tests by A A. Skochinsky on dusts from some Donets Basin seams in a test gallery of an explosives factory, and by Levitsky and N. N Chernitsin at the test tunnel of the Makeevka Central Rescue Station

If the dust is settled in the working and an igniter, for example a blank shot fired from a mortar, first makes it airborne, it is extremely difficult to determine the lower limit of explosibility of the dust because it does not completely and simultaneously become airborne Other important factors are whether the dust lies on the floor, or the roof, on the supporting timbers or elsewhere and the value of the shock needed to make the dust airborne

French experiments on the explosion of coal dust from various seams in a large tunnel showed that coal dust with a volatile content of 30 per cent explodes at 112 g/m³ while at a volatile content of 24 per cent it explodes at 225 g/m³.

In tests at the Bruceton experimental mine, USA, dust exploded at $\delta = 80$ g/m³ ($V = 30$ per cent, 80 per cent of dust passing the No. 80 sieve) when the dust was laid on planks across and along the roadway Generally mine workings may be considered dangerous when

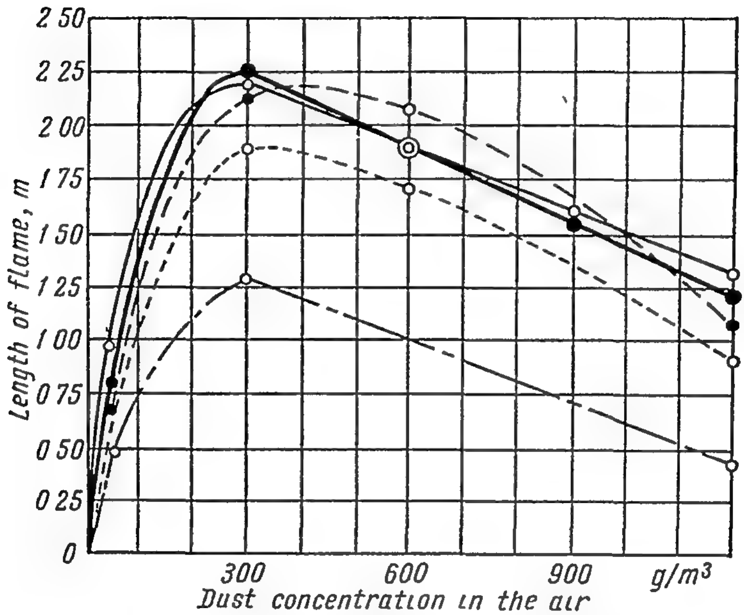

Fig. 2-10. Power of a coal-dust explosion versus dust content of the air

$\delta = 100$ to 120 g/m³. This corresponds roughly to the maximum theoretical quantity of coal dust (112 g/m³) which can burn to carbon dioxide consuming all the oxygen of the air, if we assume that the dust consists of pure carbon. However in tests in laboratories as well as in large tunnels and mines the maximum power of an explosion is achieved at a much higher dust content, e.g with $\delta = 300$ to 400 g/m³ Precisely this dust content for the explosion of maximum intensity was also determined quite definitely by numerous tests at the Makeevka Research Institute (see Fig. 2-10 showing graphs of the changes in the length of flame during coal dust explosions in relation to the airborne dust content for several Donets Basin seams). When the dust content increases further the force of the explosion drops as shown in Fig. 2-10

Excess contents of dust, as well as of methane in the air, become self-extinguishing For methane the upper limit of explosibility is not high, 14 to 16 per cent by volume or 95 to 110 g/m^3. For dust the limit is much higher and some observations showed 2,000 to 3,000 g/m^3. To draw any practical conclusions from these figures would not be correct because (a) it is extremely difficult to establish this upper limit of explosibility with any accuracy, and (b) it is always possible that in a very dusty working, the dust made airborne by the igniter may be only a part of that in the place

4 Methane The presence of methane in the air greatly increases the danger from airborne dust even at relatively low contents (1 to 2 per cent) particularly if the density of the dust cloud is below the lower limit of explosibility and the dust contains little combustible volatile matter

Indeed, if it is assumed that 1 cu m of air weighs, in average mine conditions, 1,250 g. the methane content in 1 per cent by volume corresponds to $\frac{1{,}250\times 0\ 55}{100} = 6\ 875$ g by weight, and because the calorific value of methane is 13,300 kcal/kg, the presence in the mine air of 1 per cent methane can release, in burning, about 90 kcal of heat which is roughly equivalent to $\frac{90}{7\ 5} = 12$ g of coal dust with a calorific value of 7,500 kcal/kg.

The conclusion which can be drawn is: if for example a dust begins to ignite at a density of $\delta_0 = 40$ g/m^3 in the absence of methane, then in the presence of 2 per cent of methane ignition will take place (if the methane burns) with a cloud of lower density, at $\delta_0 = 40 - 2 \times 12 = 16$ g/m^3

The significant effect of the presence of methane on the explosibility of coal dust has been confirmed by experiments in large tunnels and experimental mines and is shown by the diagrams indicating the necessary ash contents of dusts in the presence or absence of methane (see Figs 2-11 and 2-12)

For an approximate calculation of the effect of methane on the hazard from an explosion of coal dust, we can use the simple formula

$$\Delta N = \frac{100 - N}{5} \tag{2-4}$$

where N = ash content at which the dust ceases to be explosive without methane, %

ΔN = additional ash content required to render this dust non-explosive in the presence of 1 per cent methane

5 = divisor, the lower limit of explosive concentration of methane in air, 5 per cent. Thus, for example, if a dust ceases to explode at an ash content of $N = 50$ per cent without methane, then to this content must be added

10 per cent additional ash for each 1 per cent methane, if $N = 60\%$, then $\Delta N = 8\%$, etc.

The formula is deduced from the simple assumption that if the gas content in the air is 5 per cent, an explosion will take place even if the dust is 100 per cent ash. Clearly, the formula is only suitable for approximate calculations.

5 *Type and power of the igniter.* These factors are extremely important for the explosibility of the dust underground not only because the behaviour of the cloud depends on the power of the

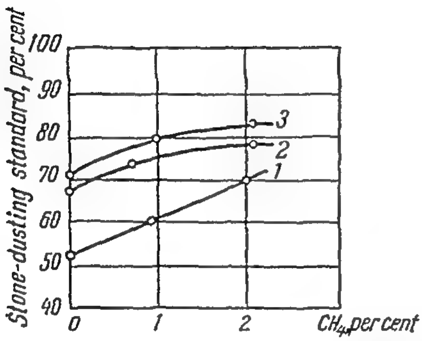

Fig 2-11 Standards of stone-dusting in relation to the methane content of the mine air

1—Kamensky seam, 2—Coking seam, 3—Seam No 3

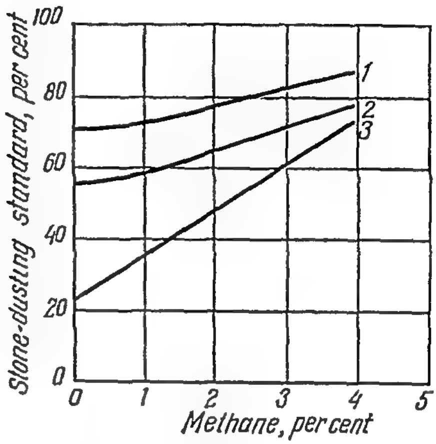

Fig 2-12 Standards of stone-dusting in relation to the methane content of the mine air for dusts from various coals·

1—volatiles 43 8%, 2—volatiles 29 0%, 3—volatiles 16 0%.

igniter (temperature, density and size of spark or flame, etc.), but also because the igniter must, as a rule, itself raise the dust, and the direction of the flame is therefore also important, i e. whether it strikes the dust accumulation directly or indirectly It has been shown by test that fine dust of $V = 25$ per cent and $\delta = 300\ g/m^3$ ceases to be explosive at ash contents of about 40 per cent if the igniter is a blank shot from a mortar, and at about 60 per cent if the igniter is intensified by some fine dust placed directly at the mouth of the mortar.

In practice therefore any conclusion drawn from tests on coal dust whether in laboratory or in test tunnel conditions can only be regarded as provisional, and deviating more or less from reality. Bearing this in mind, any test for the explosibility of dust is usually arranged so as to create more dangerous conditions than those which actually occur underground.

For example, specially fine, dry dust is tested in a cloud of optimum explosibility, powerful igniters are used, e g. blank shots from mortars charged with 1500 to 1800 g of black powder, intensified by explosions of several tens of cu m of 9 to 10 per cent methane content, etc

6. *The varying conditions of dust distribution and propagation of the explosion also affect the course of dust explosions.* The least dangerous condition of the dust generally is on the floor and sides, the most dangerous position is on the roof and on the bars of the timber props Experiments in the Bruceton mine, USA, showed that the stone dust content required to prevent coal dust exploding when it is on cross supports is 10 to 15 per cent more than when it is on the floor and sides

The presence of side workings, as well as widening or constriction of the working in which the dust has exploded exerts a considerable effect on the propagation of the explosion The composition of the wall rock is also important, i e whether it is coal, barren rock, masonry walling, etc and also whether it is wet or dry.

However it must be remarked that the effect of dampness of the working is only significant at the initial instant of the explosion because moisture makes it difficult to raise a cloud of dust, during the propagation of the explosion or in the presence of a powerful igniter, the effect of dampness is negligible Some extremely instructive experiments on the explosion of dusts from coal with 45 per cent volatiles were made in an experimental mine, the dust invariably exploded violently, despite artificial wetting to the extent of 20 per cent by weight of the dust, when[1] 83 cu m of a 10 per cent methane-air mixture was exploded at the end of the working.

2-9. CHARACTERISTIC FEATURES OF COAL DUST EXPLOSIONS

The most important features of coal dust explosions in practice are the following

1 Because of the wide variety of the properties of the coal dust formed and deposited underground, dust explosions are also extremely varied, they may be genuine explosions with dynamic and heat effects sometimes more powerful even than methane explosions, while others may be mere explosive burning with relatively slow propagation of the flame through the dust cloud, a so-called "flame in equilibrium".

2 When the working is straight and of constant cross section and more or less uniformly polluted with dust of approximately equal fineness and composition, the pressure and the speed of movement of the flame increase with distance from the focus of the explosion

In dust explosions in the large tunnel at Liévin in France, the explosions propagated at speeds from 20 to 50 m/sec at a distance of 50 m, and 100 to 150 m/sec at a distance of 150 m from the point of the dust ignition, the pressure increased from 1 5 kg/cm² at the beginning of the tunnel up to 5-10 and even 30-40 kg/cm², the maximum, at the outlet, i.e. at 200 m from the point of start of the explosion.

In experimental explosions of coal dust in another mine, pressures of 5 kg/cm² were recorded at a distance of 90 m and 8 kg/cm² at 150 m from the point of ignition, with a maximum speed of 240 m/sec.

3 The course of the dust explosion and its dynamic effects vary enormously with sharp, even though very small changes in section of the working. A more or less tranquil explosive burning of the dust cloud can be transformed instantaneously to a powerful explosion in the presence of a small obstacle such as a projecting timber or a sill This was proved, among other evidence, by special tests in a large experimental tunnel in Germany (1909-1910) The tunnel was of steel with a diameter of 2 4 m in the clear. While there was no obstacle to the propagation of the dust explosion in the tunnel the speed and pressure increased along a fairly uniform curve. But when rings of 15 × 15 cm angle iron were built into the tunnel, the force and speed of the explosion abruptly increased as can be seen from Tables 2-4 and 2-5; the pressure rose eight- to tenfold, the rate of increase of pressure was doubled or tripled although the narrowing of the tunnel was insignificant, since at the rings the diameter diminished from 2 4 only to 2.1 m.

TABLE 2-4 **Course of a Dust Explosion in an Experimental Tunnel Without Restriction but with Three Diaphragms. First Series of Tests**

Distance from point of pressure measurement to point of ignition of dust, metres	Pressure without diaphragm, kg/cm²	Pressure with diaphragms, kg/cm²	Note
90	0 32	1 61	The diaphragms were set at points 90, 105, and 120 m from the point of ignition of the coal dust
120	0 46	5 83	
130	1 12	10 50	

TABLE 2-5 **Effect of a Restriction on the Course of a Dust Explosion. Second Series of Tests**

Number and spacing of diaphragms from the point of ignition of the dust,	Highest recorded pressure, kg/cm²	Time required for the pressure to reach the maximum, seconds
One, at 60 m	0.73	1.03
Two, at 60 and 75 m	1 50	0 74
Three, at 60, 75 and 90 m	4 00	0 66

Similar observations were made in Poland from 1952 to 1954 by V. Tsibulsky studying the course of coal dust explosions in a steel tunnel of 2 m diameter and 100 m long. He established, among other things, that the speed of propagation of the explosion is a minimum if ignition starts at the beginning of the dusty part of the tunnel, and abruptly increases (up to tenfold) if the point of ignition is brought near the middle of the area.

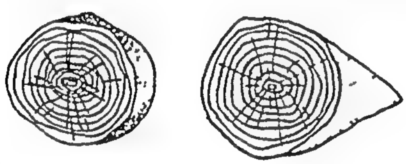

Fig 2-13 A skin (left) and a crust (right) of charred coal on props after a dust explosion

4. In dust explosions only a part of the dust burns completely, the remainder is charred and partly coked and forms characteristic skins and crusts on the timber, sides and roof of the working. A skin is a sinter of more or less coked dust and is usually oval in shape; a crust has generally not been coked at all or only very slightly and is usually triangular in cross section with a wide base sticking to the object on which it is formed Skins and crusts of dust sometimes reach thicknesses of several centimetres

Fig 2-13 shows cross sections of timbers taken from a mine in the Donets Basin where a violent coal dust explosion took place. On the timbers are traces of the flame which passed through the roadway, and skins of coked dust, the left-hand (west) side of the timber is roasted, the right-hand (east) side is covered with a dust skin. According to the investigation made after the explosion, the flame travelled along the working from west to east

During the examination of a mine for reasons of an underground explosion and in the search for the focus of the explosion, attention is always directed to these skins and crusts of dust. For a long time opinions diverged about which side of the timber the incrustation was deposited, on the side towards, or away from the flame.

The following is now considered established.

1. In areas where the flame of the explosion travelled slowly, the skin or crust is nearly always on both sides of the timbers; where the speed was high but not extreme, the skin or crust is on the side towards the flame, where the flame speed was very high the deposit is always on the side away from the flame, the side towards the flame bears only traces of the flame and cannot carry any deposit.

2. In every explosion there are two shocks the direct shock from the expansion of the gas and air, and the return shock from the contraction of the explosion products when their temperature is lowered; in addition, the presence of side roads and any type of roughness in the path of the explosion complicates it greatly. Therefore it is not possible to draw any conclusions based only on the location of the dust deposits. Other clues must also be used such as damage to the timbers, traces of flame, etc.

3. As has been pointed out several times, the formation of carbon monoxide in coal dust explosions is inevitable, and its content can reach 5-6 per cent or even more.

2-10. NATURAL MINE DUST AND DUSTINESS OF MINE WORKINGS

1. During the processes of getting coal underground there is always mixed in with the coal some rock which has been broken at the same time as the coal; natural mine dust therefore always contains, with the coal dust, some rock dust including clay shale, sandstone, limestone, etc. Mine dust therefore has a considerably higher ash content than the coal from which it comes, usually the ash content is double or triple that of the coal or even more.

While it is lying in the mine workings the dust weathers, losing some of its volatiles, absorbing oxygen, carbon dioxide and water, and becoming to some degree finer.

The fineness of mine dust is defined by its content of the —80 fraction. particles below 1/10 to 1/15 mm in size; and natural mine dusts, even from the same seam, vary widely in their content of this fine fraction, from less than 10 per cent up to 30 per cent or more, the variation between dusts from different seams can be even greater. At the face or near it the dust is most similar in composition to the coal of the same seam, and is usually of the coarser type. The ash content is highest in the haulage roads and return roadways.

N. N. Chernitsin who investigated the dust from 19 seams in the Donets Basin found that the average ash content and fineness of the dusts in the various workings were (see Table 2-6):

TABLE 2-6 **Natural Dust in the Mines of the Donets Basin, According to Chernitsin**

Location of sample	Average ash content, per cent	Average undersize of the 0 to 1 mm material through a No 80 metric screen
Production faces	8 0	25
Return airways	19 0	27
Haulage roads	34 0	65

These properties of the dust deposited underground from various sources are generally inherent not only in the Donets Basin but also in other basins

Regarding dust as material finer than 0 75 to 1 mm, the quantity of dust formed daily in a coal mine is from 0 25 to 1 or even 3 per cent of the daily output, depending on the dustiness or friability of the coal itself and of the rock, on the conditions of bedding of the seams, and on the methods of mining, breaking and transporting the coal. Particularly large amounts of dust are formed when cutter-loaders with several jibs are used

The finest dust particles, of the order of one micron or less, are carried by the air flow to the surface of the mine, but the coarser particles settle at various distances from the point where they became airborne, generally on the floor and sides of the workings and in dead ends behind the supports, in the pits and crevices of packs, etc The quantity of dust settled underground in different places varies within extremely wide limits, δ varies from a few tens of grams to several kg per cu m of a mine working

A general investigation by MakNII of coal mines yielded the following data on the dust contents of the workings in 178 seams of the Donets Basin (see Table 2-7)

2 In conclusion we should note that the dust conditions of the mine, that is the quantity of dust formed and settled in it and the properties of the dust, depend not only on the natural conditions of the seams being worked, their brittleness and bedding conditions, but particularly on the methods of mining and getting coal, which nowadays result in very high dust deposits in the mine.

Thus, for example, just a slightly dusty seam, generally not dangerous from a dust point of view if extraction is not mechanized, can become extremely dusty when it goes over to mechanized working The same can be observed when roadways which formerly were driven on a narrow face are converted to driving on a broad

TABLE 2-7 Dustiness of Air in Workings

Seams	Dust content, g/m³			
	0-100	100-300	300-600	more than 600
Number of seams	105	49	14	10
Percentage of the 178 seams investigated	59	27 5	8	5 5

face with building of packs, in which a large quantity of dust always accumulates. An abrupt change of the ventilating conditions can also increase dustiness

The practical conclusion to be drawn from this is· most careful attention must be paid to dustiness underground, and in general to the dust conditions. Continuous control is needed to prevent dust explosions, mainly by stone-dusting, that is increasing the ash content of the dust by additions of inert dust More details on the methods and techniques of the observation and control of dust and gas underground are given in special courses on the prevention of explosions of dust or gas in mines.

2-11. PRACTICABLE SAFETY LEVELS WITH MINE DUST AND METHODS OF TESTING ITS EXPLOSIBILITY

All this shows us that the degree of hazard in any particular dusty mine working, in relation to a possible coal dust explosion from a sufficiently powerful igniter depends mainly on the quantity δ, of dust accumulated, and on its explosibility, which in turn depends on its content of bituminous volatiles (V%) and finally on its fineness.

It is difficult to establish any single adequate and reliable standard for the safe condition of a dusty mine working. It is possible to say generally that when δ is less than 100-120 g/m³ explosions do not occur, but experiments show that explosions are sometimes possible even with δ less than 100 g/m³, whereas in other conditions δ must be at least 200 to 300 g/m³. Therefore in practice the decision about the safety of any particular dusty mine working is usually mainly based on the volatile content but it is difficult to establish reliable generalized standards even for this because these standards also depend on many factors including the fineness, the ash content, and the humidity, which may combine in many different ways.

On this basis, and guided by the fact that hitherto in experiments on coal dust explosions in conditions similar to those underground, the dust from seams with volatile contents below 10 to 12 per cent has never exploded even in extremely favourable conditions, Soviet Safety Regulations for coal mines and shale mines consider seams of coal with a volatile content $V = 10\%$ in the dry and ash-free condition as practically safe

If for any particular seam the value of V is above 10 per cent, the most reliable method of establishing whether or not the dust is explosive and also how much stone dust should be added to it to make it non-explosive is to test the dust from the seam in a special tunnel or experimental mine, in conditions as near as possible to those underground

2-12. TESTING OF DUST IN LARGE TUNNELS OR EXPERIMENTAL MINES AND WITH LABORATORY INSTRUMENTS

In the USSR dust tests are carried out in large tunnels at MakNII in the Donets Basin with a steel tunnel 30 m long and a reinforced concrete tunnel 250 m long, the latter being shaped like an underground roadway of about 4 sq m cross section and connected by a special stenton to a small experimental mine within the area of the research station

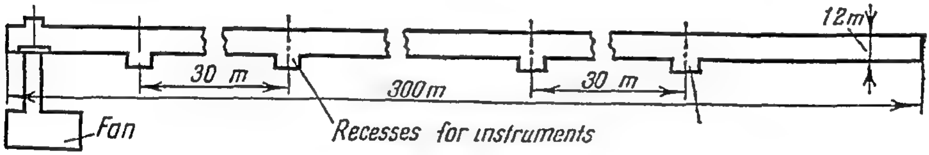

Fig 2-14 Layout of test tunnel for determining the explosibility of mine dust

Experimental metal tunnels (Fig. 2-14) are usually built of thick steel plate; they are circular in cross section, of diameter 2 25 to 2 50 m and 250 to 300 m long. Mine conditions are simulated by tests in disused mines or in specially constructed experimental mines

These tunnels and mines are fitted with powerful fans, measuring instruments, gas analysers and special mortars, shots fired from these mortars ignite the dust being tested, the dust being piled in suitable quantities on the floor of the tunnel or on planks at the tunnel supports. Natural mine dust is less explosive than artificial dust made from the same seam since they both have a higher ash content and have weathered, the dust from the seam is crushed in a ball mill or other mill until it all passes through a plus 80 sieve and all or most of it (80%) passes through a minus 80 sieve.

Besides, if the test has the purpose of establishing the amount of stone dust (inert dust) which has to be added to make the coal dust non-explosive, the experiment is carried out with the most dangerous possible content of settled dust. The stone dust as a rule should be of the same fineness as the coal dust being tested, should contain no combustible substance or in any case not more than 5 per cent, it should be dry and non-hygroscopic.

Tests in large tunnels, and all the more, those in mines, require large quantities of dust, plenty of time and high expenses. Not

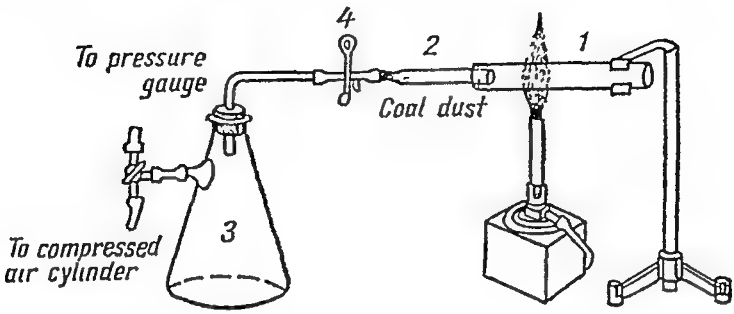

Fig. 2-15. Apparatus for laboratory testing of mine dust for explosibility

more than 2 to 3 tests can be made in a tunnel every 10 days, the dust consumed amounts to tens or hundreds of kg for each test. Tunnels are therefore generally used only for the detailed study of the dust from a particular seam of one coalfield, using this experiment as a standard of comparison for the dust from other seams, tested by simpler, usually laboratory means Large tunnels are also used for testing new methods of combating dust explosions and the statutory methods for preventing them.

Laboratory tests on dust require only a small consumption, 0.5 to 1 g per test, little time and expense. The instruments now used for these tests can be divided into three groups.

1. Gas-heated tubes, and electric furnaces, in which a small batch of dust is passed at a definite speed over a surface heated to 700-1000°C

2. Laboratory tubes The instrument (Fig 2-15) consists of two tubes the quartz tube *1*, 15 cm long, of 19 mm bore, and a glass tube *2*, 10 cm long, 10 mm bore, with one end inserted 6 to 7 mm within the tube *1*, the other end being charged with the batch of the dust being tested; a 1-litre flask *3*, filled with oxygen at a gauge pressure of 30 cm Hg, and a rubber connecting tube with a pinch *4*. The quartz tube is heated by a coal gas flame about 17.5 cm high. The essential relative positions of the flame and the quartz tube are:

the centre line of the flame must be at 5 cm from the end of the quartz tube facing the tube *2*; the tip of the inner cone of the flame must touch the lower surface of the tube *1* In these conditions the tube *1* heats rapidly enough to the required constant temperature 800°C on the inner surface of its lower wall for about 4 cm length

When the tube has heated up, the pinch is opened and a jet of oxygen flows out of the flask *3*, bringing the dust batch from the tube *2* into the tube *1*, where it passes over the heated zone and ignites more or less intensively, or does not ignite at all.

The tests are made on air-dry dust, all of which passes through a No 80 sieve The stone dust added must be standardized and of the same fineness as the coal dust. In these conditions most types of coal dust will either ignite or explode The absence of ignition shows that the coal dust is either not dangerous or nearly so If the test on pure coal dust gives a flame, the experiment is repeated to determine the degree of flammability of the dust, with additions of stone dust until ignition no longer takes place

If during the test on pure coal dust there is an explosion, indicated by a sharp sound and an intense flame, it can be estimated that the total ash content required to make this dust non-explosive will be about 50 per cent; if the flame produces a sibilant sound, from 25 to 50 per cent ash content is needed, if the flame is tranquil, less than 25 per cent

3. The laboratory tunnel is a steel tube 5.1 m long with an internal diameter of 20 cm, open at one end and closed at the other Below the first compartment is a narrower tube, 3 6 m long, joined with the upper by a series of tubes having internal flanges These flanges carry movable steel discs on which the dust to be tested is placed, during the test the dust is blown into the upper tube, acting as the test tunnel, by compressed air blown into the lower tube from a glass cylinder under an air pressure of about 6 atmospheres As the dust is blown into the tunnel a cartridge charged with 9 g of black powder is fired into the tunnel from an opening in the tail end of the tunnel.

Estimates of the explosibility of the dust being tested are based on:

(1) the length of the flame, this being measured by the burning of threads of nitrocellulose suspended from a special frame mounted in the open end of the tunnel,

(2) the amount of stone dust which has to be added to prevent ignition of the coal dust.

These instruments have been very widely used in various countries and give a correct indication of the degree of explosibility of a dust. They cannot, however, completely replace tests in large tunnels.

2-13. BRIEF INFORMATION ON THE MAIN MEASURES FOR PREVENTING COAL DUST EXPLOSIONS UNDERGROUND

The main method of preventing underground coal dust explosions is to neutralize the dust settled in the workings by additions of stone dust (inert dust), thereby increasing the ash content of the coal to such a degree that it does not ignite even at a very high temperature such as that in the detonation of explosives, or a gas explosion, an electrical spark, etc. A subsidiary method is to neutralize the dust with water*.

The whole complex of measures aimed at preventing coal dust explosions underground can be divided into 4 groups.

2-13.1 Prevention of Accumulations of Coal Dust Underground

1. The injection of the coal faces with water under pressures of 5 to 10 atmospheres or more (water infusion)**.
2. The wetting of the air in dry mines, particularly in winter, by water sprays.
3. Spraying water into the cut while it is being made by the coalcutting machine with sprays set on the jib of the coalcutter; this is also possible with added wetting agent.
4. Loading of the coal in the face without shovelling, using long chutes and shaking conveyors.
5. Plentiful watering of the coal at loading points and transfer points, and occasionally the suction of dry dust by means of exhaust fans.
6 If rail transport is used, prevention of spillage and crushing of the coal: (a) the mine cars must be in good condition without holes in the body, (b) the tracks must be in good condition, ballasted with stone; (c) full wagons and empties must be watered, both in movement and while they are being dispatched, (d) mine cars must not be overloaded.
7. Prevention of dust accumulations underground: (a) working the coal with the use of packs; (b) filling the pack surfaces with clay or guniting them; (c) wetting the dust and removing it (or sucking

---

* Watering formerly was regarded as the most reliable way of preventing dust explosions but experience has shown that mine dust quickly dries, sometimes within 1 to 2 hours, and in addition its grain size is considerably smaller after drying

** Water infusion is effective when the coal seam is not cracked but is hydrophilic, i e it is easily wetted with water. In hydrophobic coals which repel water, an addition of 1 to $1^1/_2$ per cent of a special wetting agent is needed such as naphtha soap, Nekal or sodium dibutylnaphthalene sulphonate, etc

it away) at the dustiest places, (d) periodically (3 to 4 times a year) cleaning the dust from haulage roads and return airways; (e) locating skip hoists in return airways, and not in intakes, (f) locating picking belts and dry mineral treatment plants in such a way that dust is not drawn into the mine.

2-13.2 Prevention of Ignitions of Accumulated Dust

1. Prevention of ignitions of methane.

2. During shotfiring (a) the maximum reduction of shotfiring in the coal (hydraulic bursting), (b) the use of permitted explosives, (c) observance of the rules for the limiting charges in shotholes, shotholes must not be overcharged, (d) the use of stemming. 1.5 kg of stone dust for every 200 g explosives, (e) the neutralization of the coal dust in the face and at 10 to 20 metres away from it using the following proportions

neutralizing with water $\frac{H_2O}{\text{coal dust}}$ greater than 2,

using stone dust $\frac{\text{stone dust}}{\text{coal dust}}$ greater than 3 to 4 by weight;

neutralization with ground hygroscopic salts such as $MgCl_2$, $CaCl_2$, or carnallite, $KMgCl_2$

The following methods have been successfully used for shotfiring in the last few years (a) cartridges within a water shell, (b) stemming by water contained in plastic cartridges, (c) distribution of water in special polyethylene bags containing 10 to 15 litres of water each, exploded by small electric detonators simultaneously with the shotfiring, the finely divided water acts as a fire extinguisher, (d) short-delay detonators used with a maximum delay of 130 milliseconds. According to MakNII this method is the safest and most effective way of shotfiring in fiery mines, whether gassy or dusty.

3. The prevention of coal dust ignitions during the use of electric current in the same ways as for methane ignition.

4. Lighting by reliable closed safety lamps.

2-13.3 Bringing the Settled Coal Dust in the Mine to a Safe Condition

The main statutory obligation under Soviet Safety Regulations is stone-dusting. This involves the artificial increase of the ash content of the dust settled in the workings by adding standard stone dust to it. All haulage roads and return airways are stone-dusted

on the roof and sides for their whole length if the roads are shorter than 300 m, and for at least 300 m if they are longer.

Standard stone-dusting, that is the minimum quantity of non-combustible material in a mixture of coal dust and stone-dust, has been established by MakNII or VostNII (The Eastern Mine Safety Research Station), at not less than 1 kg/m^3 of the working. The frequency of stone-dusting depends on how rapidly the coal dust settles.

The higher the volatile content of the coal, the higher generally speaking is the stone-dusting standard.

5. Watering of mine workings was formerly (35 to 40 years ago) considered the most efficient way of controlling coal dust explosions, but experience has shown the following disadvantages. (a) to be reliable the watering must continue more or less without interruption, and so much water must be provided that the dust becomes a mud, otherwise the water quickly dries out and the dried dust is finer and consequently even more dangerous; (b) the watering increases the humidity of the mine air, causing the rock to swell, sometimes even resulting in rock falls; (c) the high humidity is favourable to the development of some species of intestinal worms (ankylostoma) in the workings which may cause infectious diseases, (d) there is a very heavy expense on watering equipment. Consequently watering is nowadays used only as a supplement to stone-dusting in specially dusty places particularly in faces which are too large to be stone-dusted, also because of their steady advance and the consequent exceedingly great consumption of stone dust.

Stone dust (inert dust) must be standard and, according to Soviet Safety Regulations, must satisfy the following conditions. (a) it must all pass a metric No 12 sieve with 11 to 12 openings per cm length, and not less than 50 per cent must pass a No 75 sieve, (b) it must not contain more than 5 per cent combustible matter, nor more than 10 per cent of free silica, and in addition must contain no harmful impurity; (c) it must not cake even after a long time underground.

In addition dust must absorb as little water as possible, it must be pale in colour and should be blown up into the air as easily as coal dust.

Standard stone dust is made in special mills, usually from shale*, limestone or gypsum.

It was formerly thought that limestone dust was more efficient than shale dust because it releases CO_2 at a high temperature, and

* It must be remembered that stone dust made from shale often contains more free silica than is allowed by the Safety Regulations

that gypsum was more efficient than limestone dust because it contains about 8 per cent of water, but the latest investigations in Poland do not confirm this view.

The quantity of stone dust needed to increase the ash content of a dust up to the permitted level can be calculated from the equation

$$\frac{e+x}{1000+x}=\frac{z}{100} \tag{2-5}$$

where x = required addition of stone dust in grams per kg of mine dust

e = natural ash content of the coal in grams of ash per kg of dust

z = required ash content of the final dust, per cent by weight.

Example. If e = 100 g/kg (10%) and the required z is 65 per cent, then x = = 1,571 g or 1 57 kg per kg of mine dust

If the requirement is that z shall be 75 per cent, or only 10 per cent more than previously, then x = 2,600 g, that is the stone dust consumption is about 65 per cent more

The stone-dusting is done by hand or by wheeled stone dust spreaders of various types (ejectors working by compressed air or fans with electric drive).

The output of stone-dusting by hand is 30 to 60 m of roadway of 4 to 5 sq m cross section per manshift, and by machine up to 300 m/hour

The actual consumption of stone dust in Soviet mines is from 8 to 15 kg per m of roadway of 4 to 5 sq m cross section, and generally from 1 to 3 kg of dust per ton of daily output of the mine

The duration of effectiveness of the stone-dusting for the mine as a whole is generally 3 to 5 months, but for individual areas may be only a few weeks or days The dustiest places, near loading points, conveyors, or chutes into mine-cars, require daily additional stone-dusting In places with heavy dust settlement, according to MakNII, in addition to stone-dusting every shift it is advisable to remove the dust, and to spray it with a binding solution.

Special investigations into the stone-dusting of workings with wet stone dust in Britain showed that dry dust is better than wet dust for delaying the propagation of an explosion Preliminary wetting however makes the stone dust stick better to the walls, floor and roof of the working The speed with which the wet dust dries depends upon the rate of air flow and the wetness of the working Thus at a relative humidity of 80 per cent in a normal air current, some 1 to 3 days are needed for the dust to dry out, and at a relative humidity of 80 to 90 per cent about a week

The ash content in a stone-dusted airway is checked by eye every day and not less than once every three months by taking dust samples; not less than one sample is taken every 100 m in the gate roads near the faces and every 300 m at distances of 300 m or more from the faces.

The quality of stone-dusting in mines is determined by means of the device ПКО-1м. It is based on the ability of pure or insufficiently stone-dusted coal dust to explode when settling on a helix heated to a definite temperature. The quality of stone-dusting is checked by the length of the flame formed in the tube of the device when the coal dust is ignited by a helix heated to 1150°C.

The amount of settled coal dust is measured with the РОТОП-3, which utilizes the properties of radioactive radiations.

The measuring range of the device is from zero to 70 g/m^2, its sensitivity to dust being 1 g/m^2. The accuracy of readings is $\pm$10 per cent of the value measured

For a long time it has been considered that the stone-dusting of mine workings fully guarantees them from explosions of coal dust, but in the last decade, observations and tests have shown that in dusty, modern, highly mechanized mines, a layer of coal dust can form on top of the stone dust Within two or three hours this may be enough to create a powerful explosion if this coal dust becomes airborne and ignites. Often after the stone dust has been laid it is blown away or it cakes, and in addition the coal dust and stone dust are usually in layers. To eliminate these disadvantages of stone-dusting, MakNII has proposed the following measures, which were tested in an experimental tunnel. (a) stone-dusting not by hand but by machine; (b) the addition to the stone dust of a mixture of paraffin wax and rosin (0 5%) to prevent caking.

Recently together with stone-dusting, the method of binding the settled dust has been widely used. Binding is done by (a) water with a wetting agent, wetting by water alone being inefficient; (b) wetting with hygroscopic salts added to the water, (c) wetting by various pastes.

The wetting agent ДБ at a concentration of 0 5 per cent can be spread at a rate of 0.2 litre per sq m of the working, but its disadvantage is that the dust dries quickly.

This disadvantage is eliminated if hygroscopic salts are added to the water, for example sodium chloride or calcium chloride. The work is done in the following way the dust lying on the floor is wetted with a solution containing 0 2 per cent of the ДБ wetting agent; on this dust is poured a layer of fine common salt amounting to one third of the weight of the water, the wetted layer is lightly consolidated. Per sq m of floor of the working, 10 litres of 0 2 per cent aqueous solution of wetting agent are used, with 3 kg of common salt

Good results have been obtained by wetting the walls and the supports with various pastes; these are mixtures of water, hygroscopic liquids and a wetting agent, with 3 to 5 per cent of aluminium hydroxide or magnesium oxide to impart to the mixture the consistency of a paste About 3 kg of paste is used per sq m of the working. By comparison with the use of hygroscopic liquids, the paste is about 5 to 10 times more efficient. instead of 5 to 7 days it keeps the working safe for some 25 to 70 days.

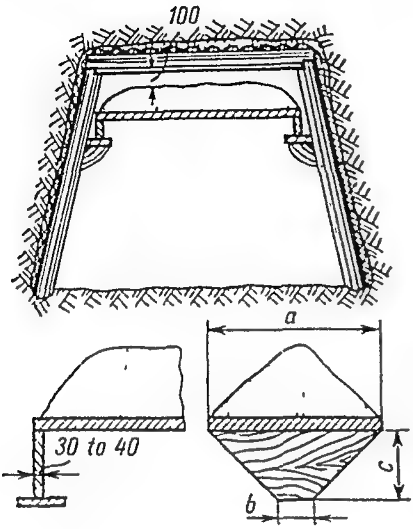

Fig 2-16 Stone-dust barrier

2-13.4 *Localization of Coal Dust Explosions*

Stone-dusting prevents the ignition of coal dust but it cannot stop the spreading of an explosion once started. For this purpose, stone-dust barriers are used; these are shelves installed usually across the working, carrying piles of stone dust When the air blast ahead of the flame of the explosion overturns the shelves, the stone dust spills in the air and forms a dust curtain which extinguishes the flame. Numerous tests of stone-dust barriers in experimental tunnels in various countries, as well as their successful use underground, have confirmed that the requirement to build them in coal mines is well founded

The amount of stone dust required in stone-dust barriers is not less than 400 kg/m^2 of the working cross section for main roads and not less than 200 kg/m^2 for subsidiary (gate) roads (Fig. 2-16) The dimensions of stone-dust barriers are given below

Width of shelf, a, mm	Width of support, b, mm	Height of support, c, mm
500	100	200
400	80	160
300	60	120

Stone-dust barriers are effective only when the explosion "races away" and therefore they are placed at a distance of some 50 to 75 m from the most probable source of a coal dust ignition. Primarily they block the roads of the district next to the faces, and those ahead of them

The efficiency of stone-dust barriers was recently studied with great care in Poland in an experimental steel tunnel 2 m in diameter and 100 m long using coal dust, 85 per cent of which was 75 microns, containing 41 5 per cent volatile matter, the stone dusts were from shale and limestone, with barriers made of planks at one level and two levels.

These tests subsequently repeated underground proved the following

1. The old belief that limestone dust is about twice as efficient as shale dust is incorrect

2 Barriers built in two layers are more effective than those built in one layer and 80 kg of stone dust per sq m of cross section was sufficient with two layers, instead of 200 kg for the ordinary barriers built in one layer.

The disadvantage of barriers built in two layers is that they block the roadway.

To increase the reliability of stone-dust barriers, two types of barriers are recommended, the main barriers described above and primary barriers. Primary barriers are designed for stopping the dust explosion in its first stage before it has become powerful. They are installed at 15 to 30 m from the face and are fitted with a device for overturning the shelves of stone dust; at this short distance the explosion itself may not have enough power to overturn them. The device consists of a photoelectric cell operated by the infrared heat radiated from the flame; the impulse to the photoelectric cell is converted into a powerful electric current which eventually fires an electric detonator operating a mechanism to overturn the shelves of the stone-dust barrier.

Water barriers. These barriers, proposed more than 50 years ago, have again aroused interest. Recent experiments in the Dortmund experimental mine in West Germany have shown that water barriers are no less effective than stone-dust barriers and are more effective against either a weak or a powerful explosion of coal dust. The recommended amount of water is 200 litres per sq m of the working cross section. The material used for the containers in West Germany is polyvinyl chloride which is semi-transparent, and consequently a light will show how full it is, it is difficult to burn, strong, and stable The capacity of the containers is 50 to 80 litres.

2-14. EXPLOSIONS OF SHALE, SULPHIDE, AND SULPHUR DUSTS

Apart from coal dust, other mineral dusts are also explosive.

Shale dust It has been shown by MakNII that shale dust, notwithstanding its high ash content, can explode. Such explosions in shale

mines have been caused by non-observance of the Safety Regulations concerning shotfiring; in particular the amount of explosive placed in the hole was much higher than permitted, the shots were fired without stemming, and the face was not sprayed before shotfiring

To prevent explosions in shale mines it is necessary to: (a) water the face until the settled dust has a water content of 50 per cent or more, (b) use stemming in the hole, preferably water stemming; (c) reduce both the weight of the explosive charge, and the number of shots fired simultaneously; and (d) if the shots are fired by fuze they must be lit one at a time.

It is advisable to use a water curtain at the shotfiring point with two or three polyethelene bags containing water

Sulphide and sulphur dusts. In the mining of sulphur and copper pyrite as well as of sulphur, explosions of the mineral dust can be very dangerous since they produce large quantities of sulphur dioxide

An explosion of these dusts can occur with an airborne dust content of 250 to 1,500 g/m^3 at a particle size of about 0 1 mm.

The airborne dusts of sulphides and sulphur are more easily ignited than coal dust and methane, coal dust ignites at 750 to 1100°C, methane at 650 to 750°C, but sulphide dust at 430 to 460°C and sulphur dust (elemental sulphur) at 275 to 340°C.

Ignitions and explosions of sulphur-containing dusts in mines are generally caused by shotfiring, the intensity of the dust explosion depending mainly on three factors the properties of the airborne mixture, the properties of the explosive, and the conditions of blasting

Mines which present an explosion hazard because of sulphur-containing dusts are divided into two groups according to the average sulphur content in the ore Group 1 from 12 to 18 per cent, Group 2 above 18 per cent.

At sulphur contents below 12 per cent in the ore, the mine is considered to be not gassy or fiery.

For mines in Group 1 the Soviet Safety Regulations prescribe the following obligatory action before blasting (a) sulphurous dust must be washed from the walls of the workings; (b) the face must be watered.

For mines of Group 2 shotfiring also must be carried out by electrical means, using permitted explosives, and any electrical equipment must be explosion-proof

Any dust formed during shotfiring may be settled by the use of curtains of water mist, created by special mist sprays. The most efficient spray for settling the dust contains mist particles of some 10 to 15 microns

2-15. COAL DUST EXPLOSIONS ON THE SURFACE AT BRIQUETTING PLANTS, OR PICKING BELTS, AND IN MINE BUILDINGS

Explosions of coal dust can take place on the surface in locations such as mineral sorting plants, briquetting works, etc., and sometimes in buildings above old mines.

The main method of preventing such incidents is periodical removal of the dust from the walls of these buildings by washing them throughout.

CHAPTER 3

THE MINE CLIMATE

3-1. GENERAL

The climate of the mine is mainly determined by the temperature and humidity of the mine air which generally differ greatly from those at the surface

The atmospheric air enters the mine at a temperature and relative humidity which correspond to the surface conditions, and, while it moves along the underground workings, suffers changes which are illustrated by Tables 3-1 and 3-2 and Figs 3-1 and 3-2.

TABLE 3-1 Temperature and Humidity (Monthly Average) of the Mine Air in the Mines of the Donets Basin

Name of mine	Distance from the downcast shaft to the face, metres	Depth, metres	Air quantity entering the mine, cubic metres per min	Month of observation	Surface air condition		Temperatures in the faces and developments, °C			Relative humidities in the faces and developments, per cent		
					temperature, °C	relative humidity, per cent	minimum	maximum	average	minimum	maximum	average
Central (Grishino)	800-1,000	157	1,095	2	−8 9	—	16 4	18 3	17 0	96	99	98
				6	+19 6	63	17 6	19 6	18 2	94	98	96
Gorlovka No 1	1,000-1,700	555	3,150	12	−2 5	93	18 0	23 0	20 4	91	98	95
				7	+22 4	57	20 9	24 6	22 9	89	97	90

The downcast shaft, and the underground roadways next to it, the pit bottom, and some of the cross-measures drifts, act as regulators for the temperature of the intake air, warming it in winter, and cooling it in summer In the haulage roads and the faces in particular the air is always heated, and the maximum temperature is found in the faces or the return airways beyond them In the return

TABLE 3-2 Mine Air Temperatures, Degrees C

Location	Yearly average	Summer (July)	Winter (February)	Yearly variation
Downcast shaft,				
collar	10	19	0	19
pit bottom	18	24	11	13
Crosscut, 950 m horizon,				
start-end	18-20	24-24	11-15	13-9
Faces, start-end	20-29	24-31	15-27	9-4
Return airway, at horizon 850 m, start-end	29-26	31-28	27-24	4-4
Upcast shaft,				
pit bottom	26	28	24	4
fan drift	18	20	15	5

airways and the upcast shaft the temperature is more or less constant throughout the year and generally somewhat lower than in the faces. The relative air humidity in downcast shafts and in roadways close to the pit bottom is generally 90 to 95 per cent, or even more if the shaft has dripping water.

Otherwise the humidity depends in part on the humidity of the surface air, but mainly on the temperatures.

In winter the atmospheric air, even if saturated with water, contains a very small absolute quantity of water vapour, and when it passes heated into the pit bottom, its relative humidity is low, and it continues to fall as it passes along the haulage roads and faces while it is getting hotter. However, if the mine workings are wet to the extent of having dripping water, the haulage roads and the faces may well have a relative humidity of 90 per cent or more even in winter. In the return airways and in particular in the upcast shafts the air is almost always very damp, of 90 to 100 per cent relative humidity. In deep mines (800 to 1000 m) the air is usually much drier than in shallow mines because the ground is drier at depth, and also because the rock is hot.

Exceptionally low relative humidity (25 per cent or less) is found in potash mines because they do not have inflows of water, and moreover potash ore is hygroscopic.

One consequence of this is that mine air in winter can have a strongly drying effect on the mine workings under appropriate temperature conditions Let us therefore consider the following

practical instance. The air enters the mine at a temperature of $t = 0°C$ and a relative humidity $n = 90\%$, leaving the mine at $t_1 = 20°$ and $n_1 = 100\%$, the air flow through the mine averaging 4,800 m^3/min. Under these conditions every cubic metre of intake air contains $4.7 \times 0.90 = 4.2$ g water and the return air contains 16.9 g water, having absorbed $16.9 - 4.2 = 12.7$ g water. The total quantity of water thus removed daily from the mine by the air is therefore.

$$\frac{12.7 \times 4{,}800 \times 1{,}440}{1{,}000^2} = 88 \text{ tons}$$

On very hot or rainy days, on the contrary, it is possible that the mine may be wetted by the air through the condensation of the water vapour from it.

3-2. FACTORS CONTROLLING THE TEMPERATURE OF THE MINE AIR

Only in shallow mines (50 to 100 m) and in small workings does the mine air temperature depend to any substantial degree on the surface air temperature. Generally speaking, the magnitude and fluctuations of the mine air temperature (daily and annual) depend on the combined action of a number of factors, of which the surface air temperature is only secondary. The most important of these factors are

1. The heating of the air by compression (auto-compression) as it passes down the shaft.
2. The rock temperature and the heat exchange between the rock and the air.
3. Various exothermic (releasing heat into the air) and endothermic (absorbing heat from the air) processes taking place underground.
4. The intensity of the air flow.
5. The temperature of the surface air.

The heating of the air as it passes down the shaft and inclined workings is caused by the compression of the air. This process is accompanied by the release of heat which raises the air temperature and can be roughly calculated by the formula

$$\frac{T_2}{T_1} = \left(\frac{P_2}{P_1}\right)^{\frac{K-1}{K}} \qquad (3\text{-}1)$$

where T_1 and T_2 = absolute temperatures of the air before and after compression

P_1 and P_2 = absolute pressures of the air before and after compression, kg/cm^2

K = ratio of the specific heats of air at constant pressure and at constant volume $K = \frac{0.241}{0.172} = 1.405$

The approximate increase of air temperature by auto-compression is 1°C per 100 m depth As the air leaves the mine through the upcast shaft it expands and the temperature correspondingly falls by about 0 8 to 0.9°C per 100 m depth.

The rock temperature for the first 30 or so metres vertical depth from the surface changes with the season of the year depending on the surface temperature, but at the level of constant annual temperature which is at 25 to 30 m depth in temperate latitudes, the temperature remains constant and close to the average annual temperature throughout the year, or about 1 5 to 2°C above it. For the southern part of the USSR the average annual temperature is from +7 to +9°C and for the northern part from +4°C to +5°C or less. In the Donets Basin it is +8° to +8 5°C.

At greater depths the rock temperature increases under the effect of the heat from the centre of the earth, and its rate of increase is described by a temperature gradient known as the *geothermal* (or *geothermic*) *gradient*, this is expressed in metres of depth for each increase of 1°C in rock temperature. Approximate average values of the geothermal gradient are: (a) for petroleum or bitumen deposits 10 to 15 m; (b) for coal deposits 30 to 35 m; (c) for ore deposits 45 to 50 m.

It must be remembered that the value of the geothermic gradient varies widely, depending on local conditions such as the rock composition, the proximity of mountain ranges which attract rainfall, river valleys, lakes, etc. Thus, for example, for bitumen deposits, known values of the geothermic gradient are 6 to 28 m, for coal deposits they are from 18 to 55 m, and for ore deposits 35 to 125 m

A systematic study of the geothermal gradient helps to forecast the temperature conditions of work in deep mines and was begun in the Donets Basin many years ago

Table 3-3 gives values of the geothermal gradient for various areas of the Donets Basin, obtained recently by S. M Bul'kach, Kraskovsky, A I Ksenofontova, A. N. Shcherban, and Ya. N Kashpur

In metal-mining regions the geothermal gradient is usually less steep than in coalfields. Thus, according to different sources, in the Krivoi Rog ironfield the geothermal gradient is about 60 m, and in the South African gold mines it reaches 120 m per degree C.

For approximate forecasts of the rock temperature at various depths, the following formula can be used·

$$\theta = t_g + \frac{H - h_1}{gr} \qquad (3\text{-}2)$$

TABLE 3-3 **Geothermal Gradient for Various Areas and Mines of the Donets Basin***

Name of district or mine	Angle of dip of seams, °C	Average annual air temperature at the surface, °C	Geothermal gradient, metres per °C	Expected temperature at 1,500 m depth, °C	Expected temperature at 1,000 m depth, °C
Makeyevka district	8-30	7 5	29 7	57	40
Gorlovka, Kochegarka Mine, central area, south wing	50-60	7 5	36 0	48	35
Artem Mine, Dzerzhinsky Coal Trust, central area, south wing	35-68	7 5	31 8	54	38
Krasny Profintern Mine, central area, south wing	60-70	7 5	35 2	49	35
Dzerzhinsky Mine, central area, south wing	45-55	7 5	33 5	52	36
Menzhinsky Mine, south-west area, Marievsky sub-area	15-18	7 9	29 5	58 5	41
Artem Mine No 1/2, Shakhtinsky-Nesvetaevsky area	7-20	8 3	38 2	47	34

* The geothermal gradient is calculated from the equation

$$gr = \frac{H - h}{t - t_{av}} \tag{3-3}$$

in which H = depth of measurement, metres
h = depth of the rock at which the temperature is constant, $h = 30$ metres
t = temperature at depth H, °C
t_{av} = average annual air temperature for the region
gr = geothermal gradient

in which θ = required rock temperature at depth H m, °C
h_1 = depth of the zone of constant annual temperature t_g, °C at the mine, metres
gr = average geothermal gradient for the area, metres per degree C

Thus, assuming an average value of gr to be 33 m, for the Donets Basin t_g will equal 9° to 10°C at 30 m depth, and we can therefore expect that at 1,000 m depth the rock temperature will be

$$\theta = 9\ 5 + \frac{970}{33} \approx 39°\mathrm{C}$$

The quantity of heat lost by the rock to the mine air depends on the temperature difference between the rock and the air, the coefficient of heat transfer of the rock, the rate of air flow, and other factors In connection with the continually increasing depth of mining and the increasing importance of the heat exchange between the rock and the mine air, an investigation into this subject was conducted recently by A N. Shcherban, A F. Voropayev, V M. Ogiyevsky, A. N. Yagelsky, and others. It has thrown some light on this complicated process but has also shown that further research is needed.

This heat exchange and the corresponding temperature change in the mine air are difficult to calculate exactly, among other reasons because the rock temperature in the airways varies with time and is not equal to the virgin rock temperature at that depth.

In shallow workings, the mine air in summer heats the rock, and in winter is heated by it, cooling the rock. At greater depths the air is ordinarily heated by the rock, cooling the walls of the roadways Consequently a zone is gradually formed around every roadway or shaft in which the rock temperature differs from the virgin rock temperature This zone which we can call the temperature-equalizing jacket, plays an important part in the heat exchange between the rock and the mine air.

The rate of formation and the thickness of the temperature-equalizing jacket depend on the temperature, rate, and quantity of the air flow through the mine, also on the virgin rock temperature at the particular depth, as well as on its thermal conductivity, and on the coefficient of heat transfer of the walls of the airway*.

According to some observations in the deep coal mines in the Ruhr, the rock around the mine airways is at first cooled rapidly, at 5 cm per day in the first few days, and then more slowly until after 1 to 2 years cooling almost stops at a depth of 12 to 20 m in sandstones, at 8 to 10 m in clay shales, and at 3 to 5 m in coal

It therefore follows that the thickness of the temperature-equalizing jacket depends also on the age of the working, and is not homogeneous throughout its length but diminishes in the direction of the air flow. At about 1,000-1,500 m from the downcast shaft, the thickness of the jacket approaches zero (Fig. 3-1)

Of the processes underground which emit heat (exothermic processes), the most important are those concerned with the oxidation of coal, the rotting of wood, and the decomposition of some rocks or minerals such as pyrite The importance of these processes can

* The thermal conductivity of sandstone is equal to $\frac{2.2\ \text{kcal}}{\text{m hr °C}}$, that of coal is $\frac{0.25\ \text{kcal}}{\text{m hr °C}}$

In addition to these most important heat processes, the mine receives heat from many other sources men, animals, lamps, machines, steam pipelines, shotfiring, the settlement of rock, inflows of water, etc A full-grown man at work gives out 150 to 300 kcal per hour, an oil lamp burning 6 g of oil per hour emits 66 kcal per hour, an acetylene lamp consuming 0 25 kg of carbide in six hours emits 184 kcal per hour, a storage lamp ($1^1/_2$ to 2 candlepower) emits only 1 85 kcal per hour In well ventilated mines these sources of heat generally have only local importance and do not affect the mine as a whole

Fig 3-2 due to A I Ksenofontova of the Moscow Mining Institute shows the temperature changes throughout the mine, as determined in deep mines in the Donets Basin during an investigation of the heat conditions in Soviet mines

Investigations of the temperature conditions underground yield diagrams of the temperature variation of the mine air, and the share of the various processes in this variation is indicated in Figs 3-3 and 3-4

The following tentative figures (Table 3-4) are based on the observations and calculations of A N Shcherban, who tabulated

TABLE 3-4 **Heat Liberated at Mines in the Donets Basin**

Depth, metres	Heat from rock, per cent	Heat from oxidation of coal and timber, per cent	Cooling of broken coal and rock, per cent	Machines and electrical power, per cent	Other heat sources per cent
900	45	32	8	9	6
1,000	49	29	9	8	5
1,100	52	26	9	8	5

the percentage share of heat from the various sources in the total heat exchange, investigated in four deep mines of the Donets Basin.

A classification of the data available on the heat processes underground results in the approximate scheme of the heat balance (see below) in a coal mine 1,000 m deep with a daily output of 2,000 tons, passing 10,000 cu m of air per minute

For a given quantity of heat released in unit time into the mine air from all known sources underground, the air temperature depends on its velocity, i e on the quantity of air passing per unit of time Intensive ventilation is therefore generally one of the most important ways of maintaining the air temperature underground within acceptable limits

Description of heat source	Percentage contribution to the total heat
Heat from the walls of airways	50
Oxidation of coal	30
Compression (and expansion) of air passing through the shafts and inclines	(±26)
Movement of water	12
Heat from men	4
Heat from machines	3 3
Settlement of rock	0 5
Shotfiring	0 13
Electric lighting	0 07
TOTAL (1% = 100 kcal per minute)	100%

3-3 PHYSIOLOGICAL EFFECT OF AIR TEMPERATURE ON MAN, AND HEAT STANDARDS FOR WORK UNDERGROUND

1. During the assimilation of food in the human organism, biochemical processes take place (metabolism) with the emission of heat Heat is ceaselessly emitted from the full-grown human body at 70 kcal per hour during sleep and 80 kcal per hour during the waking state at rest. During physical work, a man releases 250-400 kcal per hour or even more, depending on the intensity of the work.

Because the efficiency of the human body as a machine is low, averaging 15-20 per cent, the quantity of heat formed in the body during muscular work is several times more than the heat equivalent of the work done

Thus, for example, the heat equivalent R of the work T in kg-m should be equal to

$$R=\frac{T}{427}\ \text{kcal}$$

in which $\frac{1}{427}$ = heat equivalent of 1 kg-m of work. The actual amount of the added heat formed by the human body as a consequence of the work T is $\frac{R}{0\ 15}$ to $\frac{R}{0\ 20}$, that is roughly $6R$.

Observations and experiments in various countries have shown that the work done by a miner with hand tools in unit time (the power) is generally 3.5 to 4 kg-m per sec averaged throughout the shift, or about 0.05 horsepower, varying with the type of work from 0 02 to 0 07 h.p The heat equivalent of a power of 3.75 kg-m per sec is equal to $\frac{3\ 75\times 3\ 600}{427}$ = 31 6 kcal/hr and the quantity of heat

be seen from the following calculation An increase of 0.1 per cent in the CO_2 content in the mine air corresponds to an addition of 1 litre or about 2 g of this gas per cu m of air, and because the formation of 2 g of CO_2 is accompanied by the release of 4 3 kcal of heat, the temperature of 1 cu m of mine air (weighing on the average 1.25 kg, and having a specific heat of 0.24 kcal) will therefore increase by

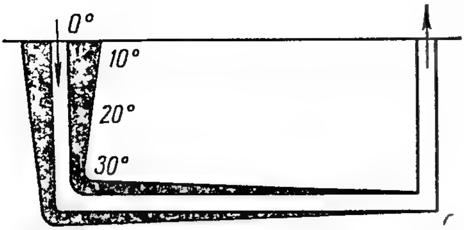

Fig 3-1 Formation of the temperature-equalizing jacket by air flow through the mine

$$\Delta t = \frac{4\ 3}{1\ 25 \times 0\ 24} \approx 14\ 5°C$$

Similarly a calculation for an increase of the CO_2 content in the mine air by 0.5 per cent caused by the oxidation of coal leads to a more than 70°C possible increase of the mine air temperature. It is known that part of the CO_2 is not formed underground but is emitted as such from the rock or coal. Nevertheless the calculations

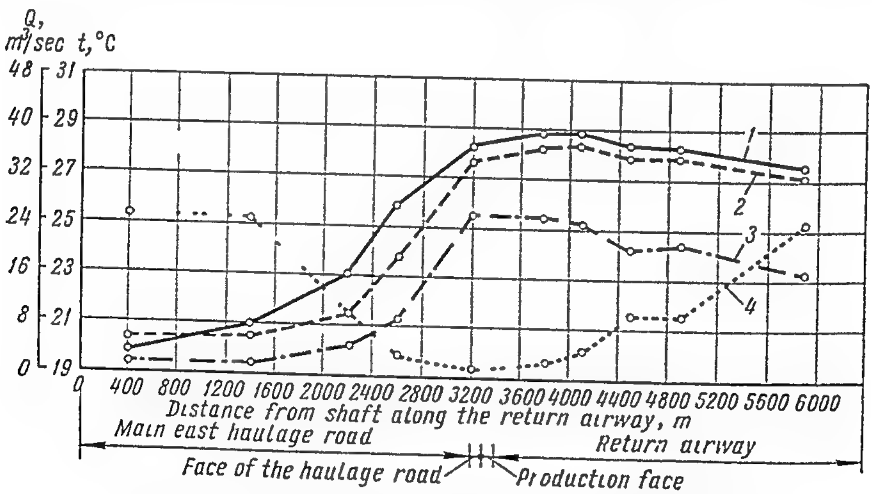

Fig 3-2 Changes in underground temperature
1—rock, 2—dry air, 3—damp air, 4—air quantity, m³/sec

do indicate that exothermic processes are very important in raising the mine air temperature They would make mining extremely difficult if they were not accompanied by endothermic processes, absorbing heat, one of the most important being the evaporation of water 1 g of water evaporating underground absorbs about 0.59 kcal of heat, reducing the temperature of 1 cu m of mine air by about 1 9°C

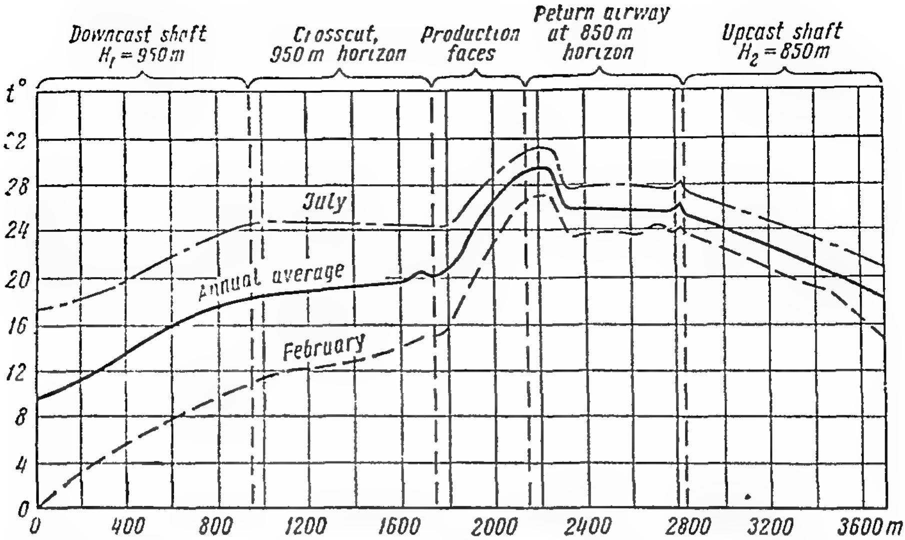

Fig. 3-3. Diagram of temperature changes in winter and summer in a deep coal mine

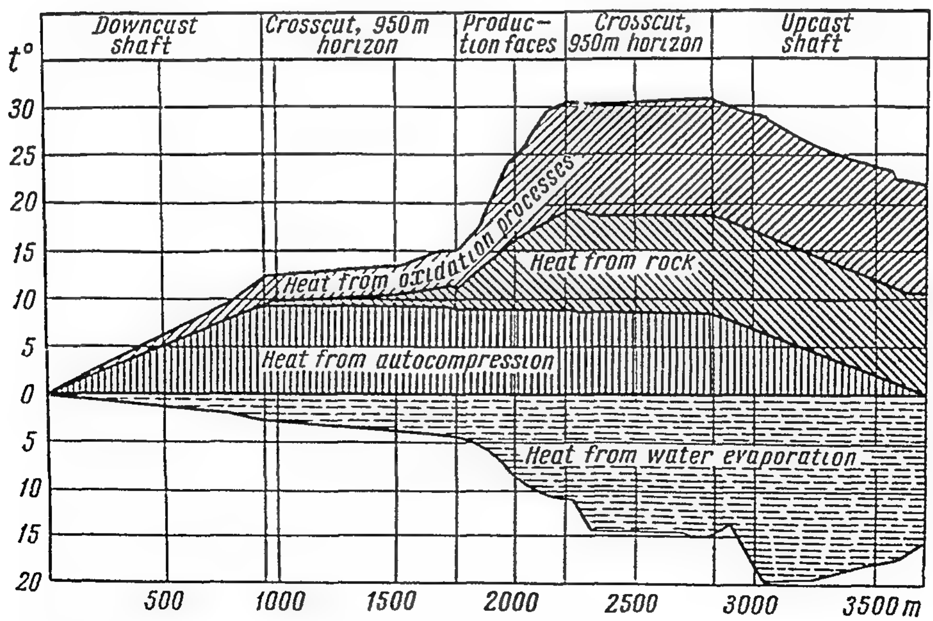

Fig. 3-4. Diagram showing the share of various processes in the changes in the mine air temperature

In addition to these most important heat processes, the mine receives heat from many other sources men, animals, lamps, machines, steam pipelines, shotfiring, the settlement of rock, inflows of water, etc A full-grown man at work gives out 150 to 300 kcal per hour, an oil lamp burning 6 g of oil per hour emits 66 kcal per hour, an acetylene lamp consuming 0 25 kg of carbide in six hours emits 184 kcal per hour, a storage lamp ($1^1/_2$ to 2 candlepower) emits only 1.85 kcal per hour In well ventilated mines these sources of heat generally have only local importance and do not affect the mine as a whole

Fig 3-2 due to A I Ksenofontova of the Moscow Mining Institute shows the temperature changes throughout the mine, as determined in deep mines in the Donets Basin during an investigation of the heat conditions in Soviet mines

Investigations of the temperature conditions underground yield diagrams of the temperature variation of the mine air, and the share of the various processes in this variation is indicated in Figs. 3-3 and 3-4

The following tentative figures (Table 3-4) are based on the observations and calculations of A N Shcherban, who tabulated

TABLE 3-4 Heat Liberated at Mines in the Donets Basin

Depth, metres	Heat from rock, per cent	Heat from oxidation of coal and timber, per cent	Cooling of broken coal and rock, per cent	Machines and electrical power, per cent	Other heat sources, per cent
900	45	32	8	9	6
1,000	49	29	9	8	5
1,100	52	26	9	8	5

the percentage share of heat from the various sources in the total heat exchange, investigated in four deep mines of the Donets Basin.

A classification of the data available on the heat processes underground results in the approximate scheme of the heat balance (see below) in a coal mine 1,000 m deep with a daily output of 2,000 tons, passing 10,000 cu m of air per minute

For a given quantity of heat released in unit time into the mine air from all known sources underground, the air temperature depends on its velocity, i e on the quantity of air passing per unit of time. Intensive ventilation is therefore generally one of the most important ways of maintaining the air temperature underground within acceptable limits.

Description of heat source	Percentage contribution to the total heat
Heat from the walls of airways	50
Oxidation of coal	30
Compression (and expansion) of air passing through the shafts and inclines	(±26)
Movement of water	12
Heat from men	4
Heat from machines	3 3
Settlement of rock	0.5
Shotfiring	0 13
Electric lighting	0 07
TOTAL (1% = 100 kcal per minute)	100%

3-3. PHYSIOLOGICAL EFFECT OF AIR TEMPERATURE ON MAN, AND HEAT STANDARDS FOR WORK UNDERGROUND

1. During the assimilation of food in the human organism, biochemical processes take place (metabolism) with the emission of heat Heat is ceaselessly emitted from the full-grown human body at 70 kcal per hour during sleep and 80 kcal per hour during the waking state at rest During physical work, a man releases 250-400 kcal per hour or even more, depending on the intensity of the work

Because the efficiency of the human body as a machine is low, averaging 15-20 per cent, the quantity of heat formed in the body during muscular work is several times more than the heat equivalent of the work done.

Thus, for example, the heat equivalent R of the work T in kg-m should be equal to

$$R=\frac{T}{427}\ \text{kcal}$$

in which $\frac{1}{427}$ = heat equivalent of 1 kg-m of work. The actual amount of the added heat formed by the human body as a consequence of the work T is $\frac{R}{0\ 15}$ to $\frac{R}{0\ 20}$, that is roughly $6R$.

Observations and experiments in various countries have shown that the work done by a miner with hand tools in unit time (the power) is generally 3.5 to 4 kg-m per sec averaged throughout the shift, or about 0.05 horsepower, varying with the type of work from 0.02 to 0.07 h p The heat equivalent of a power of 3.75 kg-m per sec is equal to $\frac{3.75\times 3\ 600}{427}$ = 31.6 kcal/hr and the quantity of heat

formed in the body when this work is done corresponds to about $\frac{31.6}{0.175}+70 \sim 250$ kcal/hr.

The total quantity of heat emitted from the grown human body per day is usually from 2,500 to 3,500 kcal, rising in exceptional cases to 9,000 kcal.

For man, as for every warm-blooded animal, the heat produced by the body must equal the heat lost If not enough heat is lost from the body, it becomes overheated with increase of the body temperature, and conversely its temperature is lowered if the heat losses are excessive In both instances not only is the normal state of the man disturbed but also his capacity for work affected

Knowing the quantity of heat (C kcal) which the body of the miner in given conditions releases every hour to the surrounding space, it is possible to calculate approximately the quantity of work (T kg-m) he can do without disturbing the normal functioning of the body, and consequently without harm to his health

It therefore follows that

$$C = 6R + 70 = \frac{6T}{427} + 70 = \frac{T}{70} + 70$$

from which the required possible quantity of work is equal to

$$T \approx 70\,(C - 70)\ \text{kg-m/hr}$$

With $C = 70$, $T = 0$, if C exceeds 70, T exceeds 0 but in order that T shall reach the normal average output of the miner (0 05 h.p), C must equal or exceed 250 kcal/hr.

All these calculations are very approximate because the human body is highly adaptable to the conditions around it, in particular it can increase or reduce its heat emission within well-known limits (thermal regulation) but it is now clear that the work capacity of the body is highly dependent on the thermal equilibrium between its heat production and its heat losses

Only a small quantity of the heat formed in the human body warms up the air and food intake, the remainder (85 to 95%) being removed to the surroundings through the body surface in three ways (1) radiation, (2) convection, (3) evaporation of sweat. The efficiency of heat release depends (a) on the air temperature, or more precisely on the temperature difference between the skin and the surrounding air, (b) on the relative humidity of the air; and (c) on the velocity of air flow

If the air is dry and hot, particularly during heavy muscular work, the miner sweats, and most of the heat is lost by evaporation of sweat; from 1 2 to as much as 3 kg of sweat can evaporate hourly, and because the evaporation of 1 g of sweat involves the loss of

0.6 kcal, it is clear that the quantity of heat whicn can be absorbed by the evaporation of this quantity of sweat is extremely large, especially if the air is in motion At an air temperature of 22 to 24°C, man has no sensation of heat or cold at rest without clothes; at higher temperatures there is a feeling of warmth, at lower temperatures a feeling of cold. If the air is completely dry, some physiologically adaptable people can remain at 120°C for several minutes, a considerably higher temperature than that at which the protein of the blood and tissue coagulates (60°C).

During the driving of the St Gotthard tunnel records were made of the highest temperatures which miners can tolerate, but only for a very short time. These figures are listed in Table 3-5.

Any prolonged stay in unfavourable heat conditions inevitably leads to overheating of the human body and an increase in its temperature; at an air temperature above 36.5°C which is the normal body temperature, the latter can rise to the dangerous limit of +42°C which may be fatal.

TABLE 3-5 **Temperatures During the Driving of the St. Gotthard Tunnel**

Degree of intensity of work	Dry-bulb air temperature, °C	Miner's condition		Dry-bulb air temperature, °C	Miner's condition	
		body temperature, °C	pulse rate		body temperature, °C	pulse rate
0	53	Rose to 39°	Rose to 125 per minute	85	Rose to 41°	Rose to 187 per minute
1	50			81		
2	46			77		
3	42			73		
4	38			69		

It is particularly distressing for a man to stay in hot, still, damp air, because the high relative humidity reduces the heat lost from the skin by sweating and the absence of air movement reduces the heat lost by convection and evaporation.

2. Thermal conditions of work underground are therefore receiving considerable attention. It has been proved theoretically and in practice that even if the air is chemically pure, man's state and his work capacity are highly dependent on his thermal equilibrium, which in turn depends on the air temperature, its relative humidity and air flow rate.

In some combinations of these factors, the state of a man doing a certain task, remains normal: he maintains his work capacity and has a feeling of comfort; in other combinations the normal state may be disturbed, the work capacity falls and there is a feeling of illness, or of "discomfort". To establish the normal heat conditions for work of a known degree of severity, it is necessary to know the various combinations of temperature, relative humidity, and air velocity, at which a man doing this work will feel comfortable.

Fig 3-5 Kata thermometer

To determine the sensation of heat from a certain combination of temperature, relative humidity and air velocity, it is not enough to measure the temperature of the air by a thermometer, the relative humidity by a psychrometer and the air velocity by an anemometer, because these instruments measure each of these factors separately, and for any particular condition their joint action must be known. The Kata thermometer is suitable for this purpose, an instrument designed by Hill (Great Britain).

3. *Kata thermometer*. The Kata thermometer measures cooling power from the joint action of temperature, relative humidity and air flow rate, at the instrument temperature of about 36.5°C, the normal temperature of the human body. It consists (Fig 3-5) of an alcohol thermometer with a cylindrical bulb at the bottom 4 cm long, having a surface area of about 22 6 sq cm, a diameter of 1 6 cm, and a hemispherical bottom

The thermometer tube is about 20 cm long and has a scale graduated from 35 to 38°C. The upper end of the stem is enlarged into an oval bulb, to receive the expanded hot alcohol, and about a quarter to a half of the volume of the bulb is then filled with alcohol The Kata thermometer is used dry or wet; when used wet the bulb is covered with a wetted muslin or cotton wick

The "factor" of every Kata thermometer in use must be known. The Kata factor of an instrument is the number of millicalories of heat which it loses per sq cm of surface area of the bulb on cooling from 38 to 35°C, and is determined experimentally from the formula

$$F=0.270\cdot S$$

where θ = temperature difference between the human body (36 5°C) and the ambient temperature, t°C

S = number of seconds required for the Kata thermometer to cool from 38 to 35°C.

The factor is determined in still air (in a calorimeter or a thermostat) at a constant temperature by observing t, S, and then calculating F. Knowing F and observing the time S, seconds, during which the instrument cools from 38 to 35°C the cooling power of the air $H=\frac{F}{S}$ is determined; this is a measure of the loss of heat by the Kata bulb in millicalories per sq cm per sec. The dry Kata indicates the heat lost by radiation and convection, the wet Kata indicates the losses by radiation, convection and evaporation. The difference between the factors for the wet Kata and the dry Kata $(H_1 - H)$ indicates the heat lost by evaporation alone*.

4. Observations have shown that at a temperature of 19°C in still air of average humidity, a man in ordinary indoor clothes sitting down quietly or doing light work in the sitting position feels a sensation of comfort (feels normal) and in this condition loses about 1.25 millical/sq cm/sec from his skin surface. The dry Kata loses 5 millical/sq cm/sec in similar conditions

The conclusion has been drawn that for the normal state of a man doing easy work the dry Kata cooling power of the atmosphere (H) should be not less than 5.

A wide investigation with the use of the Kata thermometer in various industries in Britain has yielded the following comfort standards:

	Comfort standards	
	Dry Kata cooling power, H	Wet Kata cooling power, H_1
For light work	6	18
For fairly hard work	8	25
For heavy work	10	30

* The sequence of observations is as follows The Kata bulb is lowered into water heated to 60-80°C (in a jug or more conveniently in a thermos) and is held there until the upper bulb is filled with alcohol to about one third of its volume The Kata thermometer is then removed from the water, wiped dry, suspended from a holder or a hook and watched during cooling while the alcohol column is descending When the column reaches 38°C the stop-watch is started, and at 35°C it is stopped, the number of seconds needed for the instrument to cool between these two points being noted H is the quotient of the Kata factor F and the cooling time Thus, for example, if the Kata factor is 475, and the cooling time 95 seconds, then $H = \frac{475}{95} = 5$ The determination is repeated two or three times in succession and the average is taken.

With the wet Kata the measurement is made as follows when the thermometer is removed from the water, it is lightly wiped with the fingers to remove excess water from the wick, the time S is determined and then H_1 is calculated

In use the Kata thermometer should be held as far as possible from the body and from any other hot object because it is extremely sensitive to the slightest change in temperature.

Similar work was undertaken in the Donets Basin as early as 1928-29 by the Health Inspectorate of the Artemovsk area using the Kata thermometer to investigate the work of several hundred face workers. This committee concluded that a dry Kata value of 8 can be regarded as the comfort index for face workers in the Donets Basin.

According to observations on ore fillers in the face of the South African gold mines of the Rand, their Labour productivity dropped when the dry Kata value H fell below 6 (or the wet Kata value H_1 fell below 16) as indicated in Table 3-6.

TABLE 3-6 **Drop in Labour Productivity of Ore Fillers with Deterioration of Mine Climate**

Dry Kata reading, H	6	4	2	1
Wet Kata reading, H_1	16	11.2	6.4	4
Labour productivity, per cent	100	80	60	50

Observations in a German coal mine on miners filling coal in the face showed that their productivity dropped by 23 per cent when the cooling conditions at work fell from $H_1 = 13.9$ to $H_1 = 6$ to 8. These men felt perfectly healthy at $H_1 = 15$.

Hitherto no generally accepted Kata values for the comfort of miners have been agreed, which is perfectly understandable if we remember the different ethnographical, climatic, physiological and working conditions in different countries. It must also be remembered that there is no perfect analogy between the cooling of the Kata thermometer (even if wet) and the cooling of the human body with its high adaptability.

However as a tentative conclusion on the comfort or discomfort of miners at work the figures given on page 167 can be accepted*.

* The dry Kata thermometer can also be used as an instrument for measuring air flow velocity (up to several m per min). Used for this purpose, the Kata thermometer determines first the cooling power H, and the dry-bulb temperature t of the air, the air velocity v is then deduced from empirical formulas of the type.

(a) for air velocities v above 1 m/sec

$$v = \left(\frac{\frac{H}{\Delta t} - 0.13}{0.47} \right)^2 \qquad (3\text{-}4)$$

5. Specialists in industrial ventilation have come to the conclusion that standards of comfort or discomfort can be reliable only when they are based on direct observation of the reaction of people doing various types of work, in various combinations of temperature, humidity, and air velocity. A series of experiments on many people in special laboratories resulted in the choice of a great variety of combinations of temperature, humidity, and air velocity which produced the same sensation of heat. When the results were plotted, a series of curves of effective temperature (ET for combinations in still air) and equivalent effective temperature (EET for moving air) were obtained.

By separating the maximum and minimum values of ET and EET in which the sensations of heat and well-being of individuals seemed normal, so-called comfort zones were obtained, complexes of those combinations of temperature, humidity and air velocity in which the atmospheric conditions for work were uniform and normal.

Checking of these norms of ET and EET at the Institutes of Physiology and Labour Health of the USSR showed that they needed correcting for ethnographic and climatic conditions Thus, for example, the ET comfort zone for inhabitants of the hot areas of the USSR (Transcaucasus, Central Asia) is higher, and for Central Siberia is lower than the average The characteristics of the work being done are also important As an illustration of the combinations of temperature, humidity and air velocity which correspond to comfort zones for light and heavy work, we give a summary in

---

(b) for air velocities v less than 1 m/sec

$$v=\left(\frac{\frac{H}{\Delta t}-0\ 20}{0\ 40}\right)^2 \qquad (3\text{-}5)$$

in which $\Delta t = (36\ 5 - t)°C$

The formulas 3-4, and 3-5, and the general question of the reliability and accuracy of air measurements by the Kata thermometer were investigated at the Soviet Institute for Labour Safety at Leningrad, by Maishak, as well as by Yakovenko, Ksenofontova, and others, who showed that (1) the Kata thermometer reacts sensitively to any type of eddying or other air currents in various directions and therefore indicates not only the air velocity but its general mobility, (2) the Kata factor varies with the air temperature, (3) the Kata thermometer is most accurate when it is perpendicular to the air stream, and is least accurate when the stream moves up or down along the stem of the thermometer

Appreciable inaccuracy in Kata values at temperatures near 0°C was observed by A A Ksenofontova during a survey of ventilation shafts of the Moscow region coalfield by a team from the Moscow Mining Institute

It follows that the Kata thermometer cannot be regarded as an instrument with which the air flow velocity underground can be accurately measured, but at low air velocities which are difficult to measure with the anemometer (below 10 or 15 m/min) it gives more accurate results than the so-called practical methods of air measurement used underground, such as by smoke, smell, etc.

Table 3-7 of the test results of Veller of the Moscow Institute for Labour Safety

For underground work, diagrams of ET and EET do not yet exist and it will be extremely difficult to build them up.

Under Soviet Safety Regulations the heat conditions of coal mines and shale mines are standardized in two ways by the dry-bulb temperature and the air flow velocity, subject to the strict condition that in development roads and production faces the temperature shall not exceed 25°C.

The permissible combinations of temperature and air velocity are:

t,°C	up to 15	15-20	20-22	22-24	24-25
v, m/sec	0 5 max	1 0 max.	1 0 min.	1 5 min.	2 0 min.

To establish standards of atmospheric comfort in the deepest mines of the Donets Basin, an investigation was undertaken into work both in the mine and in the laboratory This work brought out, among other conclusions, the fact that at relative humidity up to 80 per cent and at air velocities of up to 4 m per second the temperature at the end of the face could reach 27°C.

TABLE 3-7 **Combinations of Temperature, Humidity and Air Velocity, which Approximate to the Comfort Zone**

Air temperature, °C	Relative humidity, per cent	Air velocity, m per min	Remarks
20	40-50	10-15	Light work, normal state of well-being
22	40-50	20-30	
24	40-50	50-60	
26	80	80-90	
20	40-50	60-70	Heavy work, normal state of well-being
22	40-50	70-80	
24	40-50	80-90	
26	40-50	120-130	

Generally these standards are near to the comfort norms given above for the Kata thermometer but they are more convenient for the practical supervision of the heat conditions underground

In the coalfields of Europe the permitted limiting temperature for work underground is not +25°, as in the USSR, but +30° to +33°C

3-4. AIR CONDITIONING OF MINES AND METHODS OF ACHIEVING IT

Normal climatic conditions for man are therefore those in which the surrounding air is clean enough and has the requisite temperature differential to maintain the correct heat transfer rate between the body and its surroundings, sometimes for cooling the body and at other times to prevent it being overcooled.

If the heat conditions generally underground or in individual districts become distressing because the mine is deep or for some other reason, it will be necessary to restore the normal cooling power of the air and to maintain it artificially so as to avoid unhealthy working conditions and lowering the labour productivity

This is called *air conditioning*.

Air conditioning of various types for public and utility buildings and some factories by automatically maintaining the air at optimum conditions of temperature, humidity, and air velocity was begun several decades ago

TABLE 3-8 **Cooling Power of Air in Relation to Air Velocity**

Wet Kata cooling power of the air				
Wet Kata temperature, °C	Air velocity, m/sec			
	8	6	2	0 5
+20	42	33	26	16
+26	27	21	17	10
+30	17	14	10	6

Until very recently, apart from maintaining the purity of the air, air conditioning underground consisted mainly in lowering the air temperature or increasing its flow rate.

Air conditioning underground can be general, when it is applied to the whole network of mine workings or a great part of them (a horizon or one wing of a horizon) or local, involving a few districts or a few faces

At depths of 700 to 800 m, particularly in coal mines, local air conditioning is almost always sufficient and this is much cheaper and easier than general air conditioning.

At mine air temperatures of up to 25°C (wet-bulb), efficient air conditioning down to 18 to 20 degrees wet Kata can usually be obtained merely by high air velocities along the roadways Table 3-8 shows to what degree these measures are effective

Fig. 3-6 is a nomogram for approximate calculations of the increase in the air flow rate underground needed to increase the wet Kata cooling power from K to K_1.

From the nomogram, knowing the temperature and the relative humidity of the air up to its increased flow rate (or wet-bulb temperature) the required air flow rate is found at which the cooling power of the air reaches the target value.

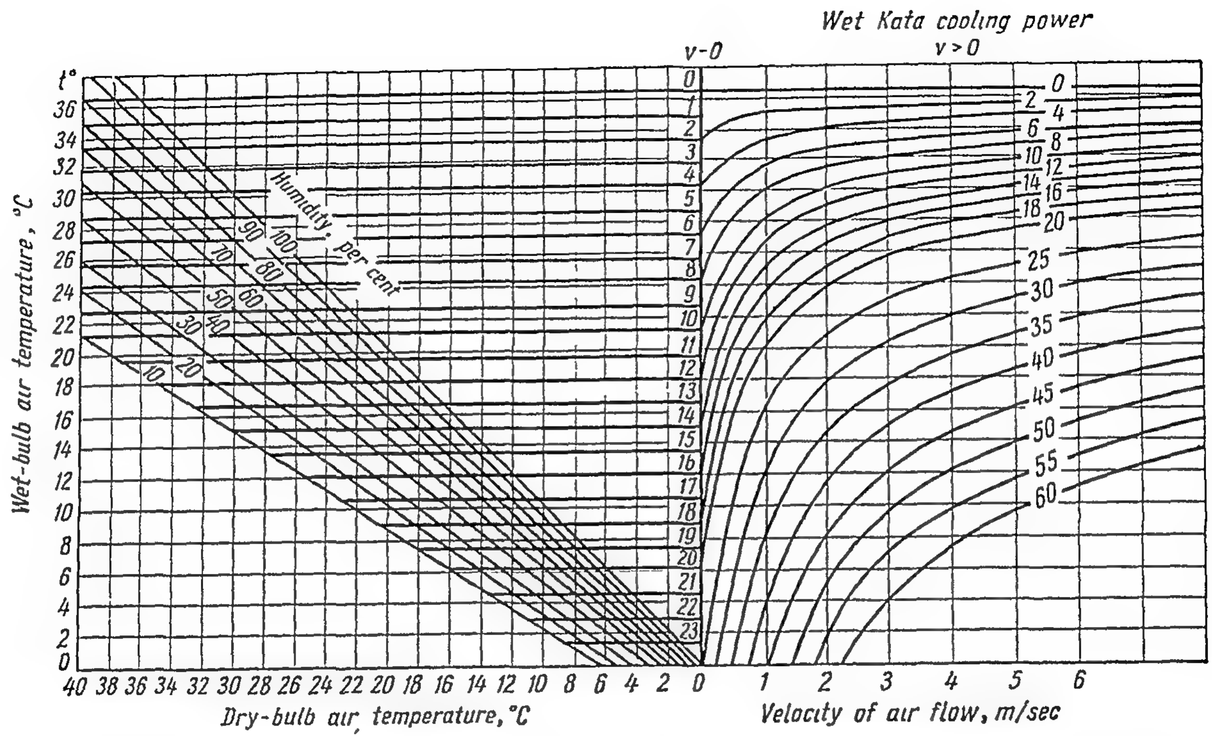

Fig 3-6. Nomogram for calculating the cooling power of the air in relation to air velocity

Thus, for example, with $t_{dry} = +28°C$ and $n = 80\%$ the necessary air flow rate to obtain a wet Kata cooling power of 25 is found as follows: from the intersection of the dry-bulb temperature with the line of humidity (left-hand side of nomogram) which corresponds to 25 degrees wet bulb, a horizontal straight line is drawn to the right to intersect with the line of cooling power corresponding to 25 Kata, and dropping a perpendicular (right-hand side of nomogram) it is found that the required speed is 5 m per second.

When it is required to determine the cooling power at a known temperature, humidity, and air flow rate, the method is as follows: from the intersection of the dry-bulb or wet-bulb temperature with the line of relative humidity a horizontal line is drawn to the right, to intersect the perpendicular line corresponding to the given velocity, and from this intersection, the required Kata cooling power for the stated conditions is found.

According to the literature on the subject, in one deep coal mine in the Ruhr, a 50 per cent increase in the air quantity passed through the mine gave the following results within 12 months. the number of faces with a dry-bulb temperature above 28°C fell from 71 to 1.3 per cent, and the average output per manshift underground increased to 35 per cent; in a metal mine near Pshibram in Czechoslovakia a 30 per cent increase in the air quantity flowing through the mine led to a 50 per cent increase in the output per manshift underground.

Similar observations were made at No. 17, 17-bis Mine in the Donets Basin at the 760 m horizon (haulage road 6.2 sq m cross section, perimeter 10.4 m) with a rock temperature of 32.5°C. The intake air parameters: $t_1 = 19.8°C$, the heat content of the air $I_1 = 11.95$ kcal/kg, the moisture content of the air $d_1 = 12$ g/kg, relative humidity $\varphi_1 = 90\%$, air quantity 502 m^3/minute, air velocity 1 35 m/sec. The values at the end of the haulage road were $t_2 = 28.2°C$, $I_2 = 19\ 02$ kcal/kg, $d_2 = 20.2$, $\varphi_2 = 90\%$.

An increase in the air flow velocity from 1.35 to 3 m/sec reduced the air temperature from 28.2 to 24.5°C, a very significant reduction, but further increases led to flattening out of the curve and even at 10 m/sec, in other words when the air velocity was more than tripled, the temperature drop was only 3°C more

Thus, increasing the air velocity was only effective up to 3 m/sec in reducing the air temperature.

The effectiveness of improvement of the heat conditions by increase of the air flow is in part explained by the fact that the quantity of heat taken from the walls of the airways increases with the increase of the air flow through them; around the airways the temperature-equalizing jacket is fairly quickly formed. This is evident

from Fig 3-7, showing the curves of reduction of air temperature in roadways driven in sandstone in a Ruhr coal mine.

One disadvantage of this method of regulating the heat conditions underground is that an increase of the air flow without a corresponding increase of the equivalent orifice of the mine is accompanied by large increases in power consumption (100% or more) and consequently in the power required of the fan installation.

The remaining methods now in use for reducing the air temperature underground can be divided into the following groups.

1 The prevention of heating of the air as it moves through the cross-measure drifts and roadways by. (a) insulation of the walls

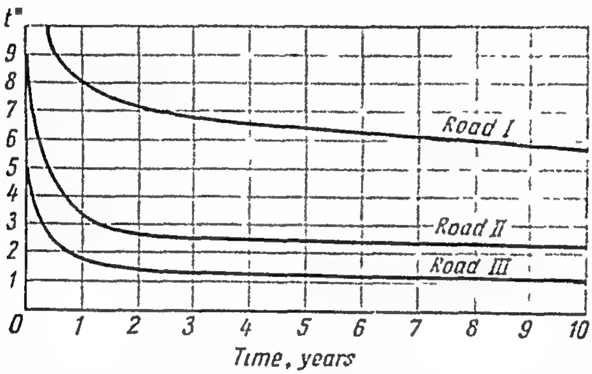

Fig 3-7 Drop in air temperature in stone drifts with age

of workings and of ventilation ducting by a jacket 20 to 30 cm thick made of sawdust, ash, and similar thermally insulating cheap material or by guniting which, however, is less efficient. This insulation is used very successfully in some Ruhr mines Thus, for example, without insulation the air entering the roadway is heated by 1 3-4°C for every 100 m length, but with the insulation the increase is only 0 3-1 8°C; (b) shortening the air circuit from the shaft to the hot district of the mine

2. Air cooling in the faces themselves (a) by the release of compressed air This means of local cooling is extremely effective*, but highly uneconomical unless the exhaust air comes from compressed-air driven machines working in the face Air should not be compressed in the face because heat is released by its compression; (b) by spraying water through special nozzles This is fairly effective since the evaporation of 1 g of water reduces the temperature of

* Thus 1 kg of compressed air, expanding freely from 6 atm to 1 atm absorbs 18 to 19 kcal and reduces the temperature of 5 cu m of air by 12°C.

1 cu m of air by about 1.75-2°C, but it requires a sufficient quantity of water under pressure and in addition increases the humidity of the mine air which is not always permissible, (c) by installing, at the face in seams up to 3 m thick, of small axial-flow fans for mixing the air and increasing the air velocity and thus its cooling power for 3 to 4 m width, a method used in smelting works and described as an "air shower".

3. The passing of the intake air through two or three dumb drifts parallel to the downcast shaft, 25 to 30 m deep, and joined to the shaft at the level of constant annual temperature Experience of some downcast shafts with these dumb drifts has shown that they can reduce the temperature of the intake air by 10°C (for example from 30 to 20°C) and that when abnormally high temperatures underground occur on hot days or during hot months, they can adequately regulate the underground temperature *

4 Keeping the mine air dry by reducing the quantity of water dropping into it by lining wet roadways, and removing the water through covered gutters The drier air has a greater cooling power

5. Artificial cooling and drying of the air in special refrigerating plants

6. Changing the mine ventilation from ascensional to descensional (when the comfort conditions underground begin to be distressing), the intake air being directed first to the upper horizons and then to the lower ones With this ventilation layout, the intake air first enters the producing districts through the upper airway, which ordinarily removes the return air, and the return air is taken away by the lower road formerly used for haulage. Because the rock in the upper horizons is cooler than in the lower horizons, with downward (descensional) ventilation the temperature of the intake air to the producing faces is some degrees (4 to 5°C) cooler. With a geothermal gradient of 30 to 40 m per 1°C, ten or more years will then pass before artificial cooling of the air will be needed, using refrigerating plant. One of the advantages of downward ventilation is that it avoids the counter-flow between coal and air in the main haulage road With this so-called homotropal ventilation the additional heating of the intake air, and its contamination by coal dust are avoided This advantage of downward ventilation has been suggested in the Ruhr coalfield and analysed for the conditions of the Donets Basin at mines No 17, 17-bis and No. 29.

The choice of the most expedient (for efficiency and economy) method of ensuring normal cooling by the mine air is reached by

---

* One well known instance is the cooling of the intake air by passing it through chambers in which ice was accumulated in winter (Canada)

from Fig 3-7, showing the curves of reduction of air temperature in roadways driven in sandstone in a Ruhr coal mine.

One disadvantage of this method of regulating the heat conditions underground is that an increase of the air flow without a corresponding increase of the equivalent orifice of the mine is accompanie by large increases in power consumption (100% or more) and co sequently in the power required of the fan installation.

The remaining methods now in use for reducing the air tempe ture underground can be divided into the following groups.

1 The prevention of heating of the air as it moves through cross-measure drifts and roadways by (a) insulation of the w

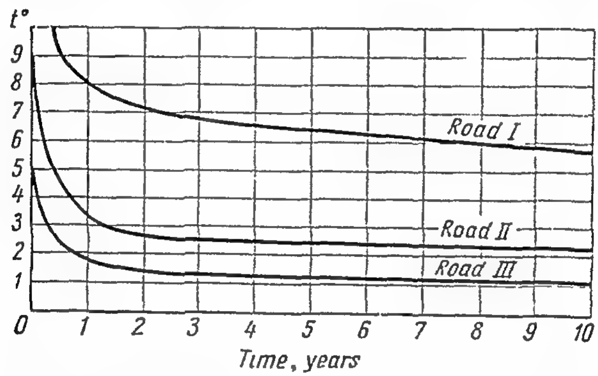

Fig 3-7 Drop in air temperature in stone drifts with age

of workings and of ventilation ducting by a jacket 20 to thick made of sawdust, ash, and similar thermally insu cheap material or by guniting which, however, is less efficient insulation is used very successfully in some Ruhr mines. Th example, without insulation the air entering the roadway by 1 3-4°C for every 100 m length, but with the insul increase is only 0 3-1 8°C, (b) shortening the air circuit shaft to the hot district of the mine

2 Air cooling in the faces themselves (a) by the rel pressed air This means of local cooling is extremel but highly uneconomical unless the exhaust air com pressed-air driven machines working in the face Ai compressed in the face because heat is released by it (b) by spraying water through special nozzles This i since the evaporation of 1 g of water reduces th

* Thus 1 kg of compressed air, expanding freely from (18 to 19 kcal and reduces the temperature of 5 cu m of air

These basic units may be physically separate depending on which of the four main types of mine air cooler are used.

1. A central installation on the surface cooling all the mine intake air.

2 A central underground installation for cooling the intake air.

3. An underground installation in the district for cooling the intake air to the district.

4 A system with the basic units in different places, for example the cooling plant and air cooler in the mine with the water cooling system on the surface, alternatively the cooling plant and the water cooler on the surface and the air cooler in the mine.

The choice of any particular layout depends in every instance on the mining conditions, the depth of the mine and other factors, the most important being mentioned below.

1. Central Cooling Installations on the Surface The advantages of this type of installation include the simplicity of handling, the ability to use a cheap coolant such as ammonia and the absence of expensive pipelines The disadvantages include the considerable losses of cooling power of the air even before the pit bottom because of the heating of the air during its compression while it passes down the shaft, particularly in shafts with much dripping water This involves the need to cool the air to temperatures close to 0°C, with sharp drops in underground temperature which can be unhealthy for the miners.

Special calculations made for the conditions of the Donets Basin and for the standards of health applied to work in Soviet mines have shown that in mines with dripping water, deeper than 900 m, cooling plants on the surface are not acceptable

At metal ore mines with a geothermal gradient 4 to 5 times as high as in coal mines there is a fair number of examples of central cooling installations on the surface.

It should also be remembered that in central refrigerating plants on the surface a considerable thickness of rock is cooled and as a result the natural ventilating draught improves Thus, for example, after the installation of the central refrigerating plant at the surface of the Oochedit mine in India, it became possible after several years to stop the fan and to use purely natural ventilation.

At Robinson Deep Mines on the Rand in South Africa, the surface refrigerating plant cools the air down to +4°C, expending 6,000,000 kcal/hr. In the shaft, which is 2,500 m deep, 35 per cent of the cooling output is lost, and another 35 per cent is lost on the way to the districts Notwithstanding these losses, the rock temperature around the main roads dropped 3°C in two years, and thanks to the increased natural draught, the quantity of intake air increased from 7,700 to 11,500 m³/min.

2 *Central Underground Cooling Plants.* The main advantage of these plants is their proximity to their objective, without loss of cooling power while the air passes down the shaft.

Their disadvantages are (1) the possibility of using cheap refrigerants is lost, (2) there are difficulties in heat removal from the condenser, because the cooling water used is the mine water which in deep mines is already hot It must therefore be artificially cooled, for example, by dropping it down an upcast shaft for 100 to 200 m

3 *Underground Cooling Plants in the Districts.* This type of plant cools the air within its own particular district and is located either at the pit bottom (like the central underground plant, but the power is lower) or close to the producing district, and may be advanced as the face advances Special calculations for the conditions of the Donets Basin have shown that for depths exceeding 900 m, district underground cooling plants will evidently be the most expedient.

4 *Cooling Plants with Separated Units* The separation of the main units of the cooling plant involves the transfer of the main cooling plant to the surface and the location of the air cooler underground, enabling cheap refrigerants to be used, eliminating difficulties with the condenser cooling and the heat removal from the electric motors, and reducing the distance between the air cooler and the face The main disadvantage of this layout is the need to provide long pipelines

The first cooling plant with separated units was built in Belgium at the Zwartberg Mine for cooling a face, the cooling plant being located in the main road, the air cooler at the face, and the cooling tower for water at the surface The technical details of this plant are shaft 1,010 m deep, angle of dip of the seam 8 5°, length of face 145 m, air required for ventilating the face 750 m^3/min, air temperature at entry to the cooling plant 32 to 34°C, temperature of the air cooled before entry to the face 14 to 16°C, temperature at the end of the face 27 to 28°C Of the total output of 350,000 kcal/hr, the face receives about 53 per cent, and the efficiency of the peant is thus relatively high.

Later a new and considerably more powerful cooling plant was installed at the same mine (2 5 million kcal/hr) for serving four more faces, and the efficiency reached 73 per cent because of several improvements.

3-5 BRIEF INFORMATION ON THE CALCULATIONS NEEDED FOR ENSURING EFFECTIVE COOLING OF THE MINE AIR

In connection with the deepening of mining and the increase of mine temperatures (in the Donets Basin there are mines already working at greater depths than 800 m and new mines are planned

for depths of 1,000 to 1,200 m where the rock temperature will be 40 to 45°C), interest in the Soviet Union has turned towards methods of calculating the necessary cooling conditions, including temperature, humidity, and flow rate of the mine air.

These investigations have thrown light on the geothermal conditions of the main Soviet mineral deposits and produced much literature on the thermal calculations for mine air.

Turning to the calculations for the practical measures aimed at ensuring normal thermal conditions of work, we must note that the movement of the mine air along the shafts and through the network of mine workings is accompanied by exceedingly complicated heat exchanges, which alter the thermodynamic parameters of the air in various ways; some of these involve heat absorption, others heat release. The complexity of the process is intensified by the fact that these factors themselves change in time and in space.

Consequently any calculations aimed at determining the values and changes of any thermodynamic parameters, as well as the heat balance of the mass of the air moving through the mine workings are extremely complicated mathematical tasks

In a general course on mine ventilation it is adequate and appropriate to limit oneself to considering merely the essence of current methods of heat calculations for mine air We will consider the analytical and in part the graphic-analytical method worked out in the USSR and checked for the actual conditions of a number of deep mines in the Donets Basin.

This method includes the solution of equations which take account of the heat emission and exchange underground subject to the following factors. (1) the heat emission from the rock, (2) the heat emission caused by the oxidation of mineral (coal, coal dust) and timber supports; (3) the heat emission from coal and rock during transport; (4) the heat emission from men, (5) the heat emission from machines, electrical power losses, and lighting; (6) the heat exchange with mine water; (7) the increase in the temperature of the air due to its compression during its movement downwards along vertical and inclined shafts, and its cooling during upward movement; (8) the heat absorbed by the evaporation of water in the wetting of the mine air, (9) the heat absorbed owing to the expansion of the exhaust air from compressed-air driven machines.

The complicated equations, some of them being integral, have been reduced by the authors to design formulas; although very bulky, they are mathematically simple and enable calculations to be made of the heat emission, heat content, and temperature of the air current as it passes through the mine. These parameters of the air flow are determined in the separate main districts, at first along the air circuit, from the surface down the shaft to the pit

bottom, along the cross-measures drift, in the main haulage road and in the face, assuming the temperature and the humidity of the intake air for midsummer conditions, the calculation is then repeated in the reverse direction, with the original temperature and air humidity values (at the upper end of the coal face) fixed according to the Mine Safety Regulations The purpose of the reverse calculation is to determine the designed temperature for the air as it leaves the air cooling plant

After the thermal conditions of the mine air current have been settled by this method, the necessary output of the cooling plant can be determined from the formula

$$Q_0 = K\,[G\,(i_1 - i_2) + Q_0'] \ \text{kcal/hr} \tag{3-6}$$

in which Q_0 = required midsummer cooling power of the plant
G = quantity of air passing through the plant, kg/hr
i_1 = heat content of the air entering the cooling plant, kcal/kg
i_2 = ditto at the outlet from the cooling plant
K = safety factor (reserve capacity) taking account of losses of cooling power in the plant (usually $K \sim \sim 1\ 1$ to $1\ 2$)
Q_0' = losses of cooling power in the pipelines of the cooling plant depending on the whole complex of its working conditions, kcal/kg.

3-6. BRIEF TECHNICAL DESCRIPTION OF A PLANT FOR LOCAL COOLING OF THE MINE AIR IN DONETS BASIN COAL MINES AT HORIZONS FROM 800 TO 1,100 M

The plant is designed for local conditioning of the air in the district haulage roads

The centrifugal cooling machines use Freon and are located underground. The condensers are cooled by circulating water which is in turn cooled by air in the return horizons

The plant is designed for cooling the air in No 9 and 12 faces of No. 17, 17-bis Mine in the Smolyaninovsky seam The haulage road of No 9 face is at 850 m depth, that of No. 12 face is at 880 m depth, the temperatures of No 9 face are +25°C below, +29°C above, with a relative humidity of 0 85 The heat conditions in summer at No 12 face are +28°C below, +31°C above, at the same relative humidity, with an air velocity of 4 to 5 m/sec and an air flow through the face of 50,000 kg/hr

The capacity of the cooling plant is designed so as to provide, while working on No 12 face alone, enough air at the top of it at

a temperature of 25°C, and while working on both faces to provide air at the upper end of both of them at 28°C. When cooling air only in No. 12 face the air leaving the cooling plant will have a temperature of 14°C and a relative humidity of 0.95, and when cooling both faces its temperature will be 19°C. The cooling power of the plant including losses will be about 600,000 kcal/hr at a boiling point in the vaporizer of +1°C, and at a boiling point of +6°C it will be 700,000 kcal/hr. The plant is local, with the air cooler in a widening of the haulage road, and the cooling plant in a room between the incline and the manway parallel to it; the water spray cooler for removing the heat of condensation of the refrigerant is in one of the return airways at the 575 m horizon near the pit bottom (Fig. 3-8).

Fig. 3-9 shows a layout with the air cooler and a water spray cooler, and Fig. 3-10 exhibits one with a surface type water cooler The plant described has a Freon cooling plant with a turbo-compressor.

When the air is cooled by a spray type cooler (Fig. 3-9) the water is forced by the pump *8* between the vaporizer *2* and the air cooler *7*. A control valve between the direct-flow and return pipes of the water circulation pipelines adjusts the flow so as to circulate less water to the vaporizer and the air cooler than to the sprays. The water filters *12* are connected in such a way that when one of them is working the other is being washed. Part of the circulating water passes through the filters to maintain it at the required level of cleanliness.

The cooling plant has two condensers *3* and *4*, connected into the circuit in turn; when one of them is working, the other one is being cleaned The condenser water circulation system also has coke filters *23*. The re-circulation pump *9* is arranged for various degrees of spraying with a constant flow of water circulating between the water spray cooler and the condenser.

The pumps *9* and *11* and the filters *23* are located on the 575 m horizon in a room on the stenton between the return airway and the intake airway. The temperature of the return air after leaving the water cooler is 32°C at 100% relative humidity

The mine water is supplied to the condensers as follows. The pump *16* takes water from the sump at the 750 m horizon and pumps it through coke filters *25* to the condensers The water heated in the condenser passes along the pipeline to the sump *19* from which the mine drainage pump *17* sends it to the shaft sump at 575 m. Make-up water for any leakages in the circulating systems is provided by clean water from the surface through a pipeline *13* with a pressure-reducing valve *14*, an automatic regulator and a tank *15* and then to the bottom of the water cooler. Clean make-up water

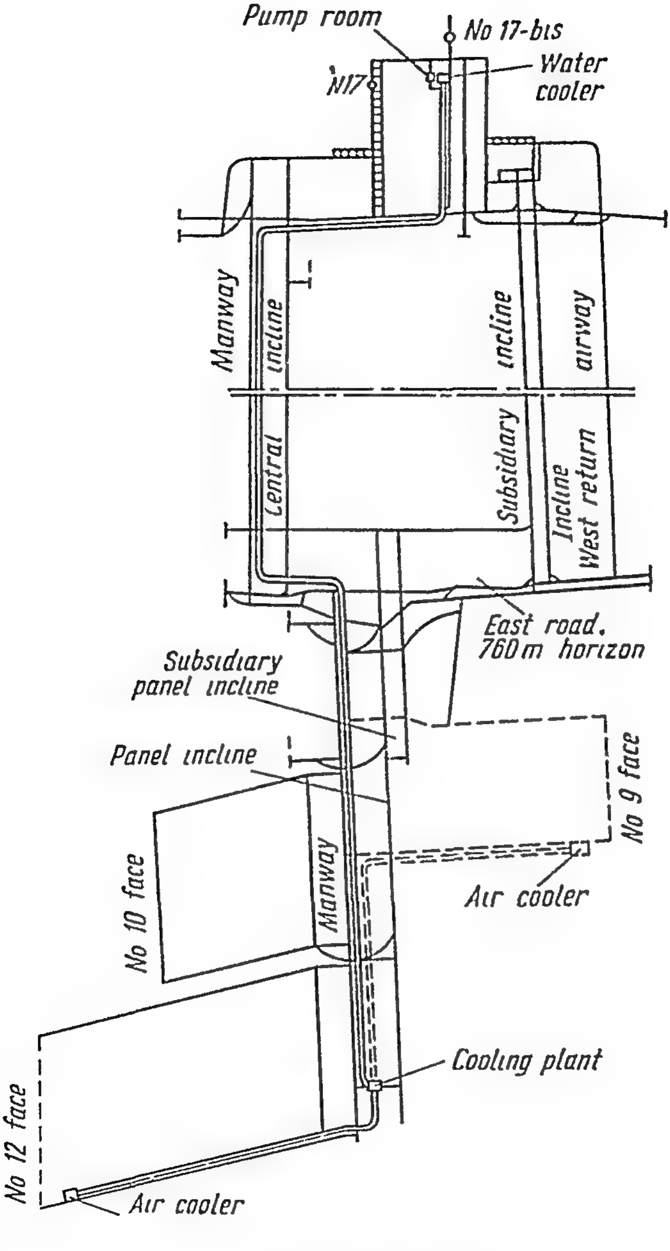

Fig. 3-8 Layout diagram of pilot underground cooling plant at No 17, 17-bis Mine in the Donets Basin

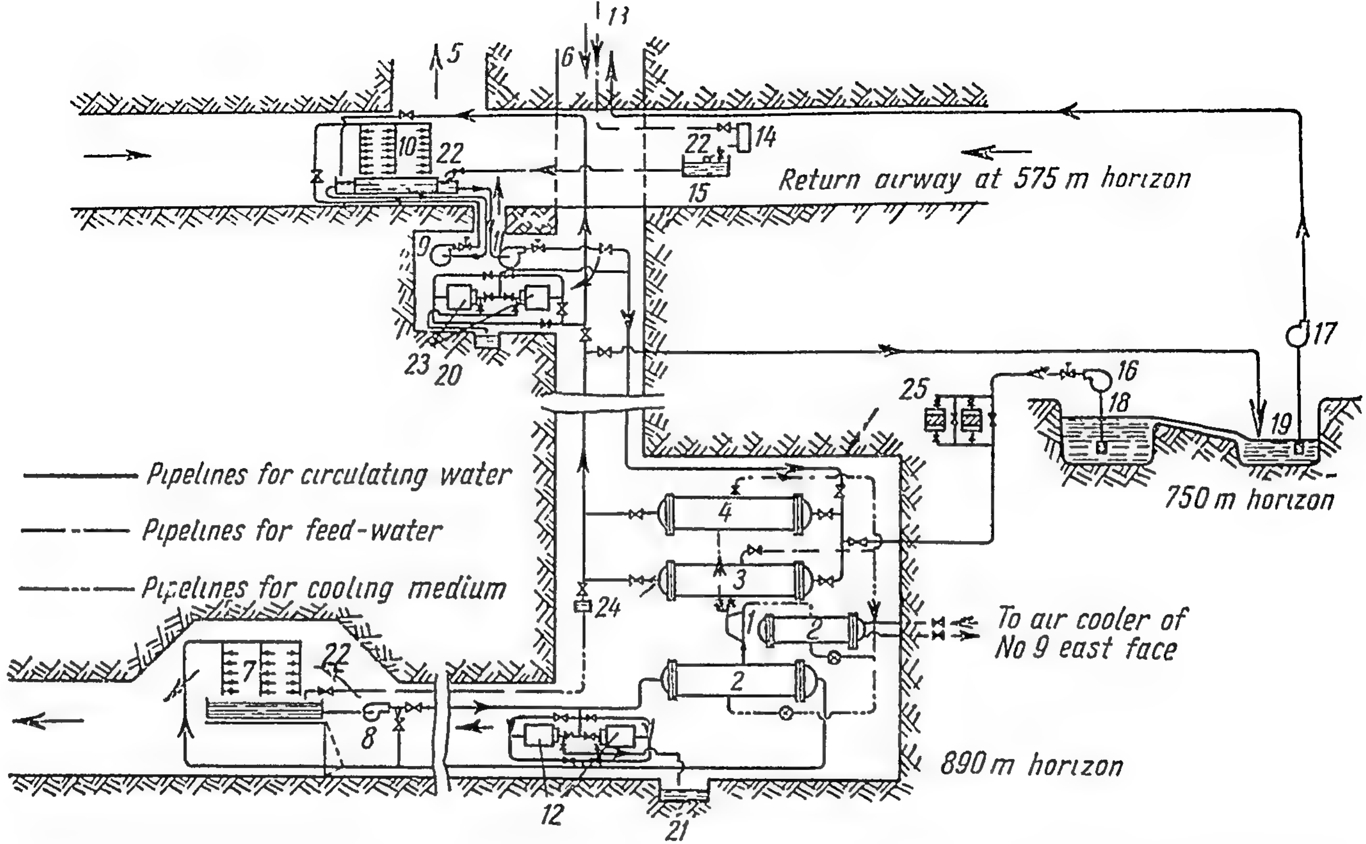

Fig. 3-9 Layout of pilot plant for cooling and drying air in deep faces at the west side of No 17, 17-bis Mine, with spray air coolers and water coolers

1—turbo-compressor, *2*—vaporizer, *3* and *4*—condensers (one stand-by, one working), *5*—upcast shaft, *6*—downcast shaft, *7*—spray cooler for air, *8*—cold-water circulation pump, *9*—re-circulation pump, *10*—water cooler, *11*—condenser-water circulation pump, *12*—coke filters for cold water, *13*—feed-water supply from the surface, *14*—break-pressure tank, *15*—intermediate tank for feed water, *16*—circulating pump for mine condenser water, *17*—mine drainage pump, *18*—sump for mine water for condenser cooling, *19*— mine drainage sump, *20*, *21*—sumps for water after washing filters, *22*—automatic regulators for feed-water consumption, *23*—coke filters for condenser water, *24*—feed-water filter, *25*—filters for mine condenser water

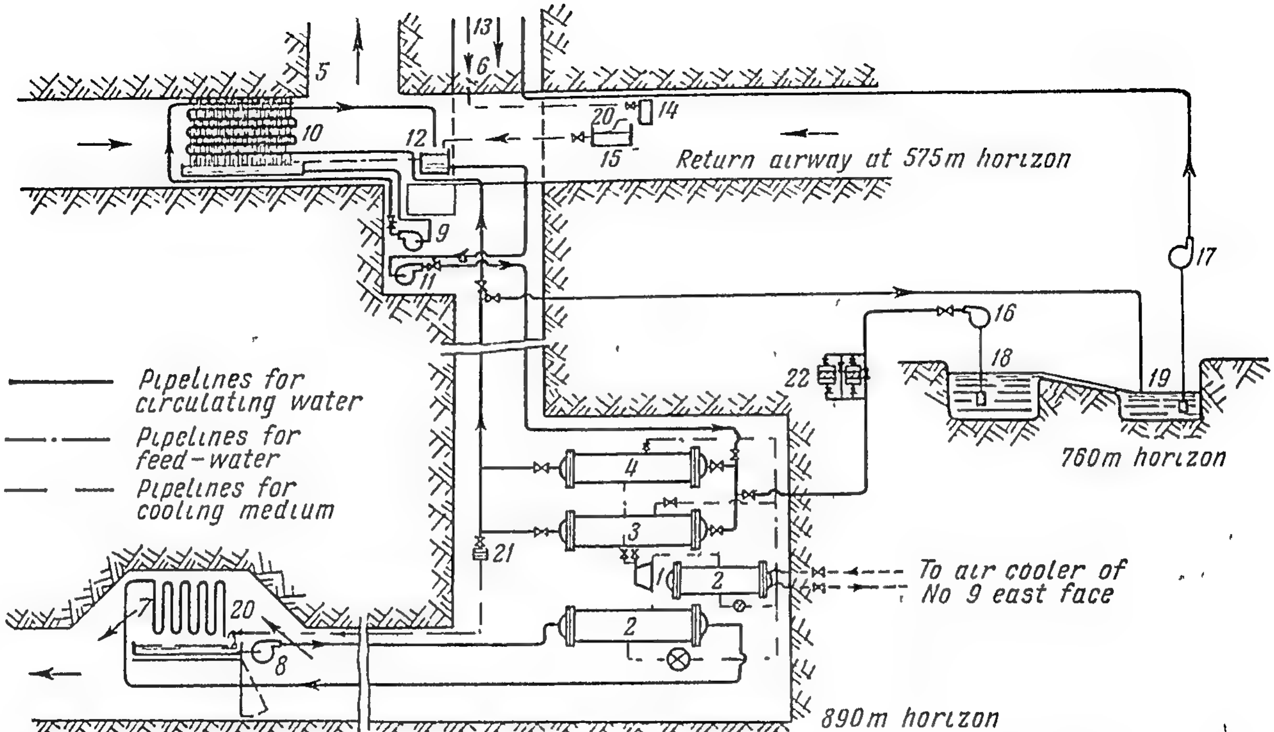

Fig. 3-10 Layout of pilot plant for cooling and drying air in deep faces at the west side of No 17, 17-bis Mine, using surface coolers for air and water.

1—turbo-compressor, *2*—vaporizer, *3* and *4*—working and stand-by condensers, *5*—upcast shaft, *6*—downcast shaft, *7*—surface air cooler, *8*—circulating pump for condenser water, *9*—pump, *10*—surface air cooler, *11*—circulating pump for condenser water, *12* — intake for pump, *13*—feed-water supply from surface, *14*—break-pressure tank, *15*—feed-water tank, *16*—circulating pump for mine condenser water, *17*—mine drainage pump; *18*—sump for mine cooling water for condenser, *19*—mine drainage sump, *20*—automatic regulator for feed-water consumption, *21*—feed-water filter, *22*—filter for mine condenser cooling water

for use in the heat transfer system is provided through the condenser circulation pipe (Table 3-9).

TABLE 3-9 **Technical Data on Cooling Plant**

Characteristics	Limits of cooling by water coolers	
	cooling one face from 28° to 14°C	cooling two faces from 28° to 19°C
Cooling power of cooling plant, kcal/hr	600,000	700,000
Designed condensation temperature of plant, °C	55	55
Diameter and maximum "round-trip" length of pipeline for cooling agent *d*, mm	125	125
L, m	2,200	4,000
Diameter and "round-trip" length for pipeline for condenser water *d*, mm	125	125
L, m	4,000	4,000
Maximum power to drive shaft, kW	394	391
Total volume of excavation required for plant, cu m	446	496

PART TWO

Mine ventilation

GENERAL

The main task of mine ventilation is to provide clean fresh air of adequate cooling power in the faces, development roads and throughout the underground workings.

This is at present done by continuously sending clean intake air underground from the surface and removing it to the surface again as dirty return air which has passed through the mine roadways.

Some of the factors which control the efficiency of mine ventilation involve its physical properties and the physico-mechanical condition of the air mass circulating underground, in particular the pressure, temperature and humidity of the air, and the velocity and conditions of the air flow.

Consequently, before describing the theory and techniques of mine ventilation we shall give basic information about the physical properties of mine air, and the laws connecting them, on the methods and devices for determining the temperature, humidity, pressure, and velocity of the air flow, and on the methods of calculating the quantity of air flowing underground.

CHAPTER 4

MAIN PROPERTIES OF THE AIR WHICH AFFECT MINE VENTILATION

4-1. PHYSICAL PROPERTIES OF AIR

1. **Mass Density.** The *mass density* ρ of air is the mass of air per unit volume:

$$\rho=\frac{M}{V}=\frac{G}{g}\times\frac{1}{V} \qquad (4\text{-}1)$$

where M = mass
V = volume
G = weight
g = acceleration due to gravity.

Since the value of g varies throughout the earth's surface, the mass density ρ is also variable. The dimensions of mass density are:

$$\rho=\frac{\text{kg sec}^2}{\text{m}^4}$$

2. **Density.** The *density* of air γ is the *specific weight* of air, or weight per unit volume

$$\gamma=\frac{G}{V} \qquad (4\text{-}2)$$

The dimensions of density are kg/m^3, consequently the density of air is the weight of a cubic metre of air expressed in kilograms. At a pressure of 760 mm Hg and at a temperature of 0°C, the density γ and the mass density ρ of dry air are respectively 1.295 kg/m^3 and 0.132 kg$\cdot$sec^2/m^4.

Since atmospheric air, and particularly mine air, always contains a certain quantity of moisture, the standard or normal density of air is taken to be 1.2 kg/m^3 which corresponds to the weight of 1 m^3 of air at a pressure of 760 mm Hg, a temperature of 15°C and a relative humidity of 60%. Correspondingly, the mass density $\rho =$ $= 0.122$ kg$\cdot$sec^2/m^4.

From Equations (4-1) and (4-2) we obtain

$$\rho=\frac{\gamma}{g} \qquad (4\text{-}3)$$

3. **Specific Gravity.** The *specific gravity* of a gas is the ratio of a unit volume of gas to a unit volume of air. For example, the specific gravity of methane is 0.554.

4. Specific Volume. The *specific volume* of air is the volume v_p (m^3) which the unit weight of air occupies

$$v_p = \frac{1}{\gamma},\ m^3/kg \tag{4-4}$$

5. Weight. The *weight* G of any volume V of air is equal to

$$G = V\ \gamma \tag{4-5}$$

6 Temperature. The *temperature* of mine air is expressed in degrees Centigrade (°C) and measured by the dry- and wet-bulb thermometers, that is, thermometers in which the mercury- or spirit-filled bulb is covered with wet gauze, muslin, or other material Because of the cooling of the bulb by the evaporation of the moisture, the wet-bulb temperature of the air which is unsaturated with water vapour is always lower than its dry-bulb temperature. In saturated air the two temperatures are the same The standard temperature used in aerodynamics and in mine ventilation is considered to be 15°C

7. Specific heat. The *specific heat* is the amount of heat (in large calories) required to raise the temperature of 1 kg of air 1°C.

The specific heat of air at constant volume (c_v) and that at constant pressure (c_p) are different With changing temperature, c_v and c_p change as follows

	—10°C	0°C	15°C	30°C	80°C
c_v	0 169	0 170	0 170	0 171	0 172
c_p	0 238	0 239	0 239	0 239	0 241

The specific heat c_p is always larger than c_v, because some additional heat energy is always spent on expanding the air. The specific heat of water vapour is 0 46

The ratio $c_p/c_v = k$ is constant for each gas, for air $k = 1.41$.

8 Viscosity. *Viscosity* is a property of air which represents its resistance to sheer stress The cause for viscosity is internal friction. According to Newton the friction between layers of a fluid is equal to

$$\mu = S \times \frac{dv}{dy} \tag{4-6}$$

where S = surface of contact of fluid layers

$\frac{dv}{dy}$ = change of velocity in a direction perpendicular to movement

μ = factor which takes account of the properties of the fluid.

Newton's formula is also valid for gases When the air is moving, the factor μ is called the viscosity of the air, its dimensions are kg sec/m^2 The viscosity of air increases with rising temperature; at 0°C and 760 mm of mercury $\mu = 1\ 71 \times 10^{-6}$.

In aerodynamics another quantity called the *kinematic viscosity* is used $\left(\frac{\mu}{\rho}\right)$ instead of the viscosity μ. The kinematic viscosity is designated by the Greek letter ν

$$\nu = \frac{\mu}{\rho} \tag{4-7}$$

The dimensions of ν = m²/sec. At 15°C and normal atmospheric pressure $\nu = 14.4 \times 10^{-6}$.

9. **Pressure.** The *pressure* of air is the physical property which matters most in all the main processes of mine ventilation. The air pressure is measured in millimetres of mercury, but the difference in pressure, because of the small value of the difference underground, is measured in millimetres of water gauge. The following types of pressure are distinguished.

The pressure of a column of air. This pressure is sometimes called the *absolute pressure.* The total force acting over the base area of a column of still air in a shaft of depth H, m, in conditions of constant density γ of the air is equal to

$$P = (p_0 + \gamma H) \cdot S$$

where p_0 = air pressure at the surface (atmospheric pressure)
S = area of the base of the column of air, m²

The pressure p per unit area of the base of the shaft is equal to

$$p = \frac{P}{S} = p_0 + \gamma H \tag{4-8}$$

and the pressure exerted by the column of air alone in the shaft is

$$p_H = \gamma H \tag{4-9}$$

The dimensions of the pressure p (also called the *unit pressure*) are kg/m². As can be seen from Equations (4-8) and (4-9) the value of this pressure *does not depend on the area of the cross section of the column of air.*

Since γ for water is equal to 1000 kg/m³, then a pressure of 1 kg/m² is equivalent to a water column of height 1 : 1000 = 0.001 m = = 1 mm, and a pressure of p kg/m² is equivalent to a water column p mm high; in other words, the pressure expressed in kg/m² is numerically equal to the value of the same pressure in mm of water.

$$p \text{ kg/m}^2 = p \text{ mm water}$$

The atmospheric pressure p_0 is usually expressed in millimetres of mercury; in order to convert this into millimetres of water gauge we must multiply it by the specific gravity of mercury which is 13.6 and, conversely, a pressure p expressed in millimetres of water

gauge is equal to $p/13.6$ mm Hg; taking account of this, Equation (4-8) becomes

$$p = 13.6 p_0 + \gamma H,\ \text{mm of water (kg/m}^2\text{)} \tag{4-10}$$

or

$$p = p_0 + \frac{\gamma H}{13.6},\ \text{mm of mercury} \tag{4-11}$$

The normal atmospheric pressure p_0 is taken as equal to 760 mm Hg or $760 \times 13.6 = 10{,}332$ mm = 10.33 m of water*.

With increasing altitude above sea level, the pressure p_0 changes approximately as follows

Altitude, m	0	100	200	300	500	800	1,000	1,200	1,500	2,000
p_0, mm Hg	760	754	742	737	716	699	674	658	635	598

The pressure increases with depth, at constant temperature, by approximately 9-10 mm Hg per 100 m of vertical depth; thus at a depth of 1000 m the pressure will be about 760 + 95 = 855 mm and at a depth of 3000 m (South African and Indian gold mines) it will be $760 + 9.5 \times \frac{3{,}000}{100} = 1{,}045$ mm Hg, that is, 12.5% and 33.5% larger than normal, respectively.

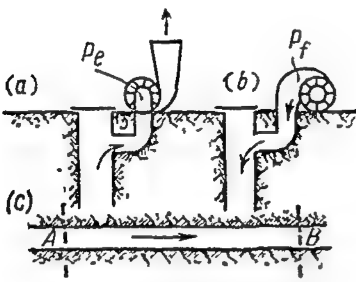

Fig 4-1 Diagram of the work of a fan at the surface of a mine (*a*, *b*); and on a length of airway (*c*)

The Artificial Pressure Created by the Mine Fan. When a fan works on a line of ducting or on the mine as a whole (Fig 4-1*a*, *b* and *c*), it creates, upstream of itself, a pressure less than atmospheric, and a pressure above atmospheric downstream; we usually say that the fan *sucks* (*exhausts*) or *blows* (*forces*) *the air The difference between the atmospheric pressure and the pressure created by an exhaust fan is called the pressure drop* (*depression*) *or head* and is denoted by the letter *h*, the term head is also used to describe the *difference between the pressure created by the fan* (*a blower*) *and the atmospheric pressure*

By comparison with atmospheric pressure a pressure drop is a rarefaction, a pressure head being an excess over atmospheric pressure

* It is worth emphasizing here that a pressure of 760 mm Hg at 0°C (762 mm at 15°C) is known as a physical atmosphere and is equal to 1.033 kg/cm² The metric or "engineering" atmosphere = 1 kg/cm² = 10 m water = 735 mm Hg at 0°C (737.4 mm at 15°C)

The value of the pressure drop or positive pressure (head) is measured, as stated, in mm of water column.

The term *pressure drop is also used to describe the pressure difference in the air between a starting point and an end point in any roadway through which air passes*

Let us denote the pressure before and after the fan as p_{ex} and p_{bl}; let the atmospheric pressure be p_0, then from the definition of the pressure drop it follows that

$$p_0 - p_{ex} = \frac{h_{ex}}{13.6} \text{ and } p_{bl} - p_0 = \frac{h_{bl}}{13.6}$$

from which

$$p_{ex} = p_0 - \frac{h_{ex}}{13.6} \text{ and } p_{bl} = p_0 + \frac{h_{bl}}{13.6} \qquad (4\text{-}12), (4\text{-}13)$$

that is, the pressure before and after the fan is equal to the atmospheric pressure, diminished by (or increased by) the value of the depression (or the positive pressure). The pressure difference of the section AB (Fig 4-1*c*) is evidently equal to $p_A - p_B$, where p_A and p_B are the pressures at the points A and B.

In courses on ventilation the pressures p_{ex} and p_{bl} are sometimes called static pressures and the corresponding gauge heights h_{ex} and h_{bl} are the static head denoted h_{st}

The Pressure Created by the Air Movement. *The pressure exerted by the movement of air on a stationary body or experienced by the front part of a moving body in still air is called the velocity pressure* and is denoted by h_v.

In hydraulics, it is proved that

$$h_v = \frac{v^2}{2g} \gamma \qquad (4\text{-}14)$$

where v = velocity of air flow, m/sec

γ = density of air, kg/m². The dimension of the velocity head is

$$h_v = \frac{v^2}{2g} \gamma = \frac{\text{m}^2/\text{sec}^2 \times \text{kg/m}^3}{\text{m/sec}^2} = \text{kg/m}^2$$

Measurement of the Static and Velocity Pressures. If we join a glass U-shaped tube half filled with water to ducting in which air is moving (Fig 4-2*a*) and the air pressure in the ducting is below atmospheric, then in the arm of the U-shaped tube which is joined to the ducting the water level will rise, and in the other arm the level will drop; the difference between the water levels in the two arms will indicate the pressure difference between the inside of the ducting and the outside of it; since the value of the pressure does not depend on the area of the water column, and each mm of water

column corresponds to a 1 kg/m² difference in pressure, any simple U-shaped tube of any diameter enables the pressure difference (or pressure drop) to be directly measured; the value of this difference in kg/m² is numerically equal to the difference in height of the water columns in mm

In the measurement of positive pressure, that is, the pressure in blown ducting, the water rises in the arm of the U-shaped tube open to the atmosphere (Fig 4-2*a*)

Let us now insert the end of a tube with a right-angle bend, in such a way that this end faces the air flow (Fig 4-2*b*). The velocity

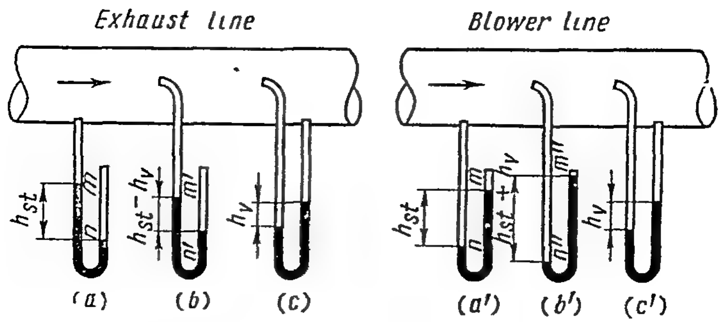

Fig 4-2 Measurement of depression and of velocity head

pressure of the air passing towards the bent end of the tube is transmitted to the meniscus in the arm joined to the ducting and the water level in this arm rises slightly less, consequently in the other arm of the U-shaped tube open to the atmosphere, the water level drops rather less, this difference in height $m'n'$ between the columns of water will be less than the first difference by the magnitude of the velocity pressure

$$h_v = \frac{v^2}{2g}\gamma$$

where v = velocity of air flow towards the bent end of the U-shaped tube

A similar test with blown ducting (Fig 4-2*b'*) shows that the difference between the levels $m''n''$ of the water columns will be larger than the difference in the heights mn by the value of the velocity pressure.

The algebraic sum of the pressure drop h_{ex}, or the head h_{bl} and the velocity head h_v is called the *total head* and is denoted by h_t.

$$h_t = h_{ex} - h_v = h_{st} - h_v \text{ in exhaust ducting} \qquad (4\text{-}15)$$

$$h_t = h_{bl} + h_v = h_{st} + h_v \text{ in blown ducting} \qquad (4\text{-}16)$$

that is, *the total pressure drop in exhaust ducting is equal to the difference between the pressure drop and the velocity head* h_v; *and the total head in the ducting under positive pressure is the sum of the positive pressure (head) and the velocity pressure* h_v; thus the velocity pressure is subtracted from a pressure drop and added to a positive pressure (head).

If both arms of the U-shaped tube are joined to the ducting as indicated in Fig. 4-2*c* and *c'*, whether the ducting is blown or under exhaust, the pressure drop or head acting on the meniscus in both arms of the U-shaped tube will be neutralized and the difference in levels between the columns of water in mm will be numerically equal to the velocity pressure h_v in kg/m².

10. Humidity of the Air. The quantity of water vapour in grams per m³ of air is called the *absolute humidity* of the air.

The higher the air temperature the larger is the quantity of water vapour contained in it. Air containing the maximum quantity of water vapour for any particular temperature is called *saturated air.*

The maximum moisture-holding capacity of the air is the absolute humidity of saturated air, it does not depend on the pressure. Appendix 1 gives the weight of water vapour per m³ of air and its vapour pressures at various temperatures

The *relative humidity* n_μ of air is the ratio of the water vapour in a given space to the amount that the space would contain if it were saturated at that temperature, alternatively it is expressed as the ratio of the vapour pressure p_v of the air to the vapour pressure of the air p_s in saturated air at the same temperature

$$n_\mu = \frac{\gamma_v}{\gamma_s} \text{ and } n' = \frac{p_v}{p_s} \tag{4-17}$$

It is not difficult to see that

$$n_\mu = n'$$

The relative humidity is usually expressed in per cent.

$$n_\mu = \frac{p_v}{p_s} \cdot 100\%$$

If the relative humidity $n_{1\mu}$ is known for the air in a given condition (p_1, t_1), then the humidity $n_{2\mu}$ for the same air in any other condition (p_2, t_2) is found, as explained in courses on physics, from the equations·

$$n_{2\mu} = n_{1\mu} \cdot \frac{p_2}{p_1} \times \frac{p_s'}{p_s''} \tag{4-18}$$

or

$$n_{2\mu} = n_{1\mu} \frac{t_1}{t_2} \times \frac{\gamma\gamma_v'}{\gamma\gamma_v''} \qquad (4\text{-}19)$$

where p_s' and p_s'' = saturated vapour pressures at temperatures t_1 and t_2

From Equations (4-18) and (4-19) it follows that, other conditions being equal, the relative humidity increases with pressure, that is, with depth, and diminishes with temperature.

The *moisture content* of the air is the amount of water vapour in kg expressed per kg of the dry part of the mixture of air and water vapour, it is denoted by the letter d (or D) Under ordinary conditions the moisture content is less than the absolute humidity of saturated air

$$d = \frac{622 \times n_\mu \times p_s}{p_0 - n_\mu \times p_s} \text{ g/kg} \qquad (4\text{-}20)$$

or

$$D = 0\ 001d \text{ kg/kg} \qquad (4\text{-}21)$$

where p_0 = atmospheric pressure, mm Hg

The *heat content* of the mixture of air and water vapour is the total amount of heat contained in the mixture per kg of the dry part, it is denoted by the letter i:

$$i = 0\ 24t_d + (595 + 0.45t_d)\, D \text{ kcal/kg} \qquad (4\text{-}22)$$

Example The measurements made are $t_d = 21°C$, $t_w = 19°C$, $p = 826$ mm Hg

Solution The relative humidity according to the table (Appendix 2) is 82%, the absolute quantity of moisture according to the table (Appendix 1) is $0\ 82 \times 18\ 2 = 14\ 9$ kg/m³, $p_s = 18\ 7$ mm Hg, the moisture content is equal to

$$d = \frac{622 \times 0\ 82 \times 18\ 7}{826 - 0\ 82 \times 18\ 7} = 11\ 7 \text{ g/kg}$$

$$D = 0.0117 \text{ kg/kg}$$

The heat content is equal to

$$i = 0\ 24 \times 21 + (595 + 0.45 \times 21) \times 0\ 0117 = 11.11 \text{ kcal/kg}$$

Evaporation of water. In mines where water runs down the exposed surfaces, and the air is not saturated with water vapour, the water continuously evaporates, but evaporation is accompanied by the absorption of heat, and therefore it reduces the air temperature

The quantity of water evaporated underground varies in wide limits from 0.2-0.3 g/m³ of air per minute in dry mines to 3-5 g/m³ in wet mines for every 100 metres It depends mainly on the air velocity, and its temperature and humidity.

Condensation of water vapour. When unsaturated air is cooled below the temperature (the dew point) at which the water vapour contained in it is the saturating value, the surplus water falls out of the air in the form of mist, dew or frost. We shall denote the quantity of water vapour per m³ of the saturated space by w_1 at the temperature t_1, and w_2 at the temperature t_2, t_2 being smaller than t_1. During the cooling of the air a quantity of water is precipitated out of it, equal to

$$q = w_1 - w_2 \tag{4-23}$$

In addition, because of the reduction in volume caused by cooling, an additional quantity of moisture q' is precipitated, courses on physics explain that if, during the process, the pressure also changes from p_1 to p_2, then

$$q' = w_2 \times \frac{p_2 T_1 - p_1 T_2}{p_2 T_1} \tag{4-24}$$

The quantity q' being small is usually neglected

Condensation of moisture underground in the form of mist, water, frost, or ice takes place

(a) in intake airways driven in rock at low temperature, particularly in permafrost areas;

(b) in upcast shafts,

(c) in fan drifts

Moisture is condensed in the form of "sweating' on the walls of roadways where self-heating of the mineral has evaporated some moisture; this moisture is condensed from the air when it comes into contact with the cold walls of the roadway.

"Sweating' is one of the signs of the start of an underground fire caused by self-heating of the mineral.

Precipitation of moisture from the mine air. Moisture can be precipitated in the form of mist, in drops and in the form of frost (ice)

Mist is formed sometimes in the mine surface buildings over the downcast shaft, in the pit bottom and within the downcast shaft, by the interaction of the warm air from the heating plant at the shaft with the cold air coming down the shaft; from the cooling of the air entering the pit bottom and striking its cold walls (in the northern parts of the USSR) and from the condensation of the water vapour as the warm return air rises up the upcast shaft. If the shaft surface buildings are made airtight and heated (Part 3, Section 13) it is possible to avoid the formation of mist at the pit bank, as well as the freezing of the cage keps, and men are less likely to catch cold

The prevention of mist in the pit bottom roadways is difficult; the basic solution would be to dry the intake air, which in mining conditions is barely acceptable; the moisture content of the intake

air can also be reduced by preventing leakages of water into the air heating ducts and plant and into the shaft; it is also possible to reduce slightly the temperature of the heated intake air entering the shaft.

Precipitation of moisture as rain also takes place in upcast shafts

The formation of frost or ice on the walls of airways occurs

(a) in summer in the northern areas of the USSR, when air comes down workings which have been cooled in winter, for example, inclined shafts,

(b) in winter, also in northern areas, in the pit bottom when the air is not heated enough; this can be prevented by increasing the heating of the air and by installing small electric heaters in places where the ice is formed,

(c) in the fan drift; this can be prevented by reducing the leakage of air through the collar of the upcast shaft, by heating this air; or by thermally insulating the fan casing if the fan is installed outside

4-2. THE MAIN PHYSICAL LAWS OF GASES AND VAPOURS

Boyle's law (Mariotte's law):

$$\frac{p_2}{p_1}=\frac{V_1}{V_2}=\frac{\gamma_2}{\gamma_1}$$

or

$$p_1V_1=p_2V_2=\text{constant} \tag{4-25}$$

from equation (4-25) we obtain

$$\gamma_2=\gamma_1\cdot\frac{p_2}{p_1} \tag{4-26}$$

Gay-Lussac's law

$$V_2=V_1(1+0.00366t) \text{ or}$$

$$\frac{V_1}{V_2}=\frac{T_1}{T_2} \tag{4-27}$$

The equation of state

$$\frac{p_1V_1}{T_1}=\frac{p_2V_2}{T_2}$$

or

$$pV=RT \tag{4-28}$$

in which for dry air $R = 29\ 27$

Dalton's law. *The total pressure of a mixture of gases and vapours is equal to the sum of the partial pressures which each of them would have if it were alone.*

Thus the partial pressure of the oxygen of the air at atmospheric pressure (760 mm mercury) and at a content of 20.93 per cent is equal to

$$760 \times \frac{20\ 93}{100} = 159 \text{ mm mercury}$$

4-3. CALCULATION OF THE DENSITY OF AIR

The specific weight, or density of air, is needed for solving many problems met with in mine ventilation.

We possess no instrument which can directly measure the density of air. To determine the density it is necessary (with a thermometer and a barometer) to measure the temperature t, the air pressure p, and to substitute these values in the following formulas.

From Equations (4-25) and (4-28) we obtain

$$\frac{1}{V_2} = \gamma_2 = \gamma_1 \cdot \frac{p_2 T_1}{p_1 T_2} \qquad (4\text{-}29)$$

If we assume $\gamma_1 = 1.2$ kg/m³, $p_1 = 760$ mm mercury, and $T_1 = 15° + 273° = 288°$, then, ignoring the subscripts, we have

$$\gamma = 0.455 \frac{p}{T} = 0.455 \frac{p}{t+273}^{*} \qquad (4\text{-}30)$$

From Equation (4-27) we can obtain another formula for gamma·

$$\gamma = \frac{p \times 13.6}{RT} \text{ kg/m}^3 \qquad (4\text{-}31)$$

Equations (4-29) and (4-31) show that the colder the air and the greater the pressure, the denser is the air. As the air passes down the shaft its temperature usually rises but at the same time its pressure increases; in this way two opposing factors thus work on the density at the same time. It will be shown below that if the temperature increase per 100 m depth, the so-called temperature gradient, does not reach 3 4°C, then the pressure effect is the greater, that is, the density of the air increases, but if the temperature increase exceeds 3.4° per 100 m, the temperature increase has the greater effect; in other words the air becomes lighter. The evaporation or condensation of moisture which usually occurs in the shaft somewhat changes this critical value of the temperature gradient.

The density of air underground generally does not deviate by more than 6-8 per cent from the standard density of 1.2 kg/m³;

---

* At temperatures below —7°C more accurate results are obtained from the formula

$$\gamma = 1\ 293 \frac{273p}{760T} = 0\ 465 \frac{p}{T}$$

however, in individual instances this deviation can be considerably higher; thus, for example, at an elevation of 2,000 m above sea level, where $t = +10°C$, the density is only 0.966 kg/m³; in a mine 1,000 m deep (below sea level) at the same temperature it will reach 1 392 kg/m³

Since the specific volume of air is inversely proportional to its specific weight, any change in the specific weight of air is accompanied by a change in its volume· the lighter the air, the greater is its volume and vice versa

To calculate the specific weight of damp air, the following equation is demonstrated in courses of physics

$$\gamma = 0\ 465 \times \frac{p}{T}\left(1 - \frac{0\ 378 \times n_{\mu} \times p_s}{p}\right) \text{kg/m}^3 \qquad (4\text{-}32)$$

All the pressures are expressed in mm of mercury; n_{μ} is the relative humidity expressed as a fraction.

Equation (4-32) shows that *the wetter the air, the lighter it is*; but if the humidity increases as a result of evaporation of water, which is a process in which heat is absorbed and consequently the air temperature is reduced, then the air density, notwithstanding the increased quantity of water vapour, will be increased rather than reduced

Because of its unwieldiness and the slight effect of the moisture content on the density of air, Equation (4-32) is not usually used, and Equation (4-30) is used instead, which takes no account of the change in humidity

Equation (4-33) below is more accurate within the following limits p from 700 to 840 mm, t from 0 to 30°C, and n_{μ} from 0.60 to 1 0, subject to the condition that the maximum error of measurement of p is ± 0.25 mm and that of t ± 0 2°C.

$$\gamma = 0.461 \times \frac{p}{T} \qquad (4\text{-}33)$$

in which the error from neglecting the change of n_{μ} does not exceed ± 1 per cent within the stated limits.

4-4. PHYSICAL CHANGES IN AIR

The physical changes in air as it moves through the mine, that is, changes in temperature and pressure, are extremely varied, it can be noticed that these changes generally conform more or less to the following processes

1 *Isochoric* (*constant volume*) *process* The temperature and pressure change, but the density and the specific volume remain constant.

2. *Isothermal process* During this process the temperature remains constant. t = constant; and the isothermal equation applies

$$pv = \text{constant} \tag{4-34}$$

3. *Adiabatic process.* When the changes of state of air occur under this law, no heat is taken from or added to it, that is $\Delta Q = 0$ (Q being the amount of heat). The adiabatic equation applies

$$pv^{1.41} = \text{constant} \tag{4-35}$$

4. *Polytropic process.* In this process the connection between the pressure and the volume is expressed by the equation

$$pv^{n} = \text{constant} \tag{4-36}$$

in which n is usually between the limits of 1 and 1.41.

However, when air expands, giving out heat, or when it is compressed and absorbs heat from outside (for example, in contact with the heated walls of a shaft), n is larger than 1 41; the same is possible when the air is compressed and its temperature rises and the heat is removed (for example, when the air passes down a shaft with water falling in it), n then being less than 1 0

If the values of pressure and temperature are known for two states of the air, then the value of the index n can be found from the equation

$$n = \log \frac{p_2}{p_1} \cdot \left(\log \frac{p_2}{p_1} - \log \frac{T_2}{T_1}\right) \tag{4-37}$$

With a linear relationship between t and H

$$n = \frac{1}{1 - Ra} \tag{4-38}$$

in which a = increase of temperature per metre; if a is larger than 0 0342°C, then n is negative. Equation (4-38) is due to A F. Voropayev

Observations on the changes of air temperature in shafts have shown that at depths below 50 m, the changes in t do not deviate much from a linear law

CHAPTER 5

MEASUREMENT OF THE TEMPERATURE, HUMIDITY, PRESSURE AND VELOCITY OF AIR

The information in this chapter, concerning instruments for measuring air pressure and the velocity of air flow underground, is confined to that which would be useful for executive engineers The more detailed information needed for scientific investigations into mine ventilation can be found in special textbooks But since for the staff of the mine ventilation service, measurements of air velocity and calculations of air quantities are daily tasks in highly varying conditions, an appropriate section of this chapter gives some details about them

5-1. MEASUREMENT OF THE TEMPERATURE OF MINE AIR

Thermometers used for measuring the mine air temperature should be housed, so as to prevent breakage, in a metal case from which they are removed when the reading is taken. Thermometers are carried in a leather case on a shoulder strap. The scale divisions vary from 0.5 to 0 2°C Temperatures are also measured with the *psychrometers* described below

Before recording the thermometer reading, the observer must be convinced that he has measured the true air temperature, and for this reason two readings are taken at an interval of 5-10 minutes they should be the same To accelerate the measurement procedure the case is attached by a short string to the thermometer and the latter is twisted for two to three minutes on the end of the string

In wet places, water must not drip on the thermometer bulb; the bulb must be wiped dry before every measurement.

5-2. MEASUREMENT OF THE ROCK TEMPERATURE

The rock temperature is measured in short or long boreholes. If the purpose of the measurement is to obtain the virgin-rock temperature, the measurement should be made in an area which is neither cooled nor heated, in the face of a development road driven narrow away from any production faces A N Shcherban considers that the holes need to be only 1 5-2.0 m deep

The following methods of measuring the temperature are used:

1. By the use of ordinary thermometers or maximum- or minimum-reading thermometers placed in a drill hole. It has been proved that some two hours after the end of drilling and taking the drill steel out of the hole, the walls of the hole which have been heated by drilling have cooled down to the original temperature The collar of the hole is closed Six to 12 hours are enough for the thermometer to take the temperature of the surrounding rock The thermometer is taken out of the borehole and quickly read.

2. By the use of a tube filled with water and placed in the borehole for not less than four hours (preferably 24 hours) after which the tube is extracted and the water temperature is taken quickly, within two minutes.

3. By the use of electrical resistance thermometers

To increase the certainty of accurate measurement, the air temperature should be measured in the same borehole at various depths, so as to draw a graph of the change of temperature with depth. Gas emission within a hole can lower the temperature, while the oxidation of coal can raise it.

5-3. MEASUREMENT OF THE HUMIDITY OF THE AIR

Humidity measurements of the air underground are usually made with the *sling psychrometer*, or with the *ventilated psychrometer*. The sling psychrometer (Fig. 5-1) consists of two thermometers mounted side by

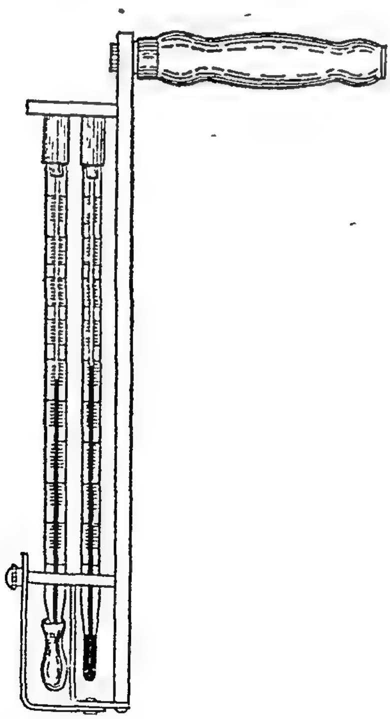

Fig 5-1. Sling psychrometer

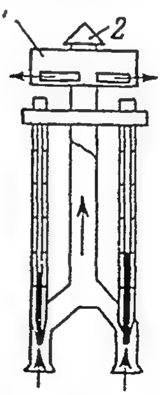

Fig. 5-2. Ventilated psychrometer
1 — fan, *2* — clockwork winding key

side on a rigid frame with a handle; the handle is used for rotating the instrument In the ventilated psychrometer (Fig 5-2) the thermometers are fixed in a metal mounting; in the upper part of the instrument is a small fan, driven by a spring, which draws air in at a certain speed through a tube beside the thermometer bulbs One of the thermometer bulbs of each of these instruments is a wet bulb covered with a wick of wet muslin net. The relative humidity n is determined from the difference in the readings between the wet- and dry-bulb thermometers using the table in Appendix 2 (when the two readings are equal, the relative humidity is evidently 100%) The rotation of the sling psychrometer continues until the readings of both thermometers become constant, usually after a half to two minutes When the thermometers are read, the frame and particularly the thermometers must not be touched with the hand, the wick must be heavily wetted but water must not drip off it The thermometer readings must be taken quickly and the lamp should be placed behind the thermometer in underground conditions

The humidity of the air can also be calculated from the equation

$$n_{air} = \frac{p_v}{p_s} = \frac{p_s - 0\ 5\,(t_d - t_w)}{p_s} \times \frac{p_0}{755} \tag{5-1}$$

where p_0 = barometric pressure
t_d = dry-bulb temperature
t_w = wet-bulb temperature, the other symbols being as on page 195

5-4. MEASUREMENT OF AIR PRESSURE

5-4 1 Measurement of the Absolute Pressure

The absolute pressure is generally expressed in mm of mercury It is usually measured underground by the aneroid barometer, occasionally by a recording barometer Soviet aneroid barometers have scales with mm and half-mm divisions from 600 to 790 mm

Barometers must be calibrated before use and supplied with a calibration certificate The certificate should carry the following corrections

(a) The *scale correction* necessitated by the elastic properties of the materials of which the aneroid is made, this correction compensates the elastic after-effect of the barometer only if the instrument is installed in one particular place, but if the barometer is moved from one place to another and in particular if it is moved up or down along a shaft or an incline this correction will be inadequate and the aneroid readings even after this correction will still be wrong.

(b) The *temperature correction* required by the effect of change of temperature on the barometer needle; to reduce the effect of abrupt temperature variations the barometer should be kept in its case; before taking a reading the barometer should be left 10-20 minutes to take up the temperature of the surrounding air

(c) A *supplementary correction* required for any inaccuracies which remain after these scale and temperature corrections have been made.

The technique of taking barometer readings is as follows. The barometer case is placed horizontally and opened and without taking the barometer out of the case it is turned so that the viewing angle is perpendicular to the point of the needle If the reading is taken underground with a miner's lamp, care must be taken that the shadow of the needle falls vertically on the scale. The barometer glass is then lightly tapped with the finger until the needle ceases to move, and the reading is taken at the end of the needle with the naked eye or preferably with a magnifying glass

Recording barometers, called barographs, record, on graph paper, the changes of atmospheric pressure; within the drum is a clockwork mechanism. If the barograph pen draws a thin line, its accuracy is about ± 1 mm Hg. Mine barographs are generally not calibrated, they record only the changes of the barometric pressure and not the absolute pressure. This is enough for checking changes of the barometric pressure

5-4.2 Measurement of Pressure Differences

The aneroid barometer can measure not only the absolute pressure of the air but also pressure differences The pressure is first measured at one point and the barometer is then carried to the other point and a second reading is taken. The difference between the two readings is equal to the water gauge depression, subject to the strict condition that both points are at the same altitude. This method of measuring pressure differences has the following disadvantages.

(1) The barometer must be carried from one point to another; during transport the barometric pressure can change considerably and consequently the measured difference in pressure will not be the true difference

(2) Most barometers give only coarse indications since their least scale division is 0 5 mm Hg which is 6.8 mm of water. However, barometers do exist, for example, the Casella type with a scale division of 0.1 mm of mercury, which are quite suitable for making pressure surveys

Other instruments are also used for underground measurements of pressure differences, for example, the ordinary U-shaped tube water gauge and micromanometers for scientific investigation.

1. **The ordinary manometer or water gauge** in the form of a glass U-shaped tube (Fig. 5-3). The tube is provided with scale divisions at mm intervals on the glass or written on a board to which the tube is fixed. To measure a pressure the tube is filled with water* to the 0 division on both sides at the middle of the scale; the two legs of the U-shaped tube are then joined by rubber tubing to the points for which the pressure difference is required, after connecting up the instrument, the water rises in the leg which is connected to the point with the lower pressure, and falls in the other leg The difference in water levels in the two legs will be numerically equal to the pressure difference:

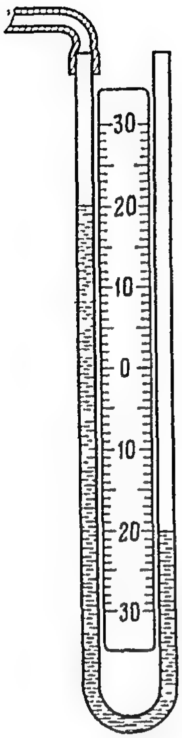

Fig 5-3 Simple water gauge

$$h \text{ mm} = \Delta p \text{ kg/m}^2$$

The readings are taken on both meniscuses and added If it is desired to read on one meniscus only, it is possible to do so by changing the scale so that the numbers on it are twice their previous value.

In a water gauge connected to the fan drift, the meniscus usually fluctuates violently, which makes the instrument difficult to read; dampers are then usually inserted, for example, small cotton wool pads placed in the rubber tubing, or short capillary tubes placed between the water gauge and the rubber tubing.

The U-shaped water gauge is one of the simplest instruments for measuring differential pressures; subject to the conditions that the legs are vertical and the divisions on the scale or on the tube itself are correct, the readings are completely reliable. If the instrument is filled incorrectly with water, for example, if the meniscus is not at the zero point when the U-tube is disconnected, the final result will nevertheless be correct although the readings on one leg will be high and on the other leg will be correspondingly small

The correct method of taking water gauge readings at the fan drift is discussed in Chapter 8

2. **The recording water gauge.** For the continuous recording of the mine ventilating pressure, mine fans are provided with recording water gauges Figure 5-4 shows the instrument made by the Kharkov Factory for Instrumentation, of ring balance type The main part

* At temperatures below 0°C, tinted kerosine is used instead of water, but the scale must be accordingly altered.

of the instrument is a hollow thin-walled annulus *1* provided with a cross-piece *2*, carrying knife-edge bearings (not shown on the drawing); with these knife-edges the annulus bears like a balance on stationary steel pads. Weights *7* on the cross-piece balance the annulus and a limiting weight *3* on the annulus is provided to limit the measurement To limit the angle of rotation there is a bearing plate *4*, and a stop *5*. The ring is half filled with liquid (2.4 litres) through an opening with a bung *6*. At the top of the ring is a shaped plate *8*, designed to transfer the movement to the writing mechanism. In its upper part, the ring is divided by a partition into two sections which are connected by tubes to the "static" and "velocity" connections of Pitot tubes. Under the effect of a pressure difference between the air in the two strips, the water in them begins to move towards the side of lowest pressure, disturbing the equilibrium of the ring and rotating it slightly. The rotation of the ring is transmitted to the pen which rises or falls and draws a line on the paper on the drum *9*. The instrument can function at air velocities, such as those in the fan drift, from 2.5 to 20 m/sec

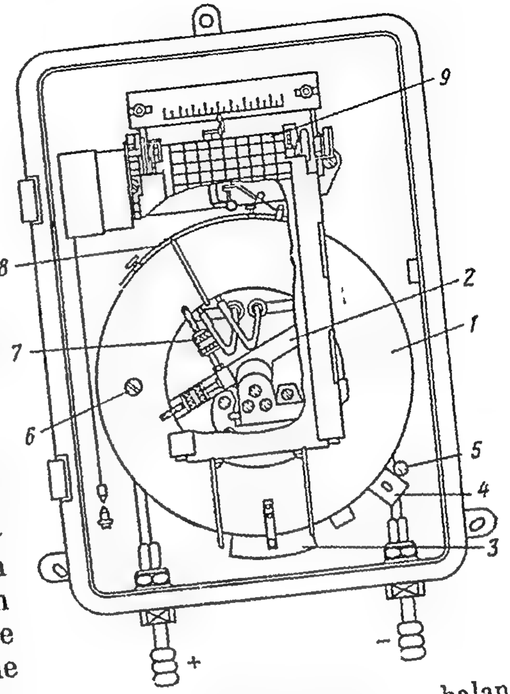

Fig 5-4 Recording (ring balance) water gauge

1—hollow thin-walled annulus, *2*—cross-piece, *3*—limiting weight, *4*—bearing plate, *5*—stops, *6*—bung, *7*—weights for statically balancing the instrument, *8*—plate for transferring the motion to the recording mechanism, *9*—chart for recording readings

Having tested this instrument in the laboratory and underground MakNII has stated that its accuracy is equal to ±2% of the measured value

Instruments for recording the quantity of the air flow are built on the same principle; when they are installed the discharge coefficient must first be determined (see p. 225)

Recording water gauges operated by clockwork are built:

(a) filled with mercury and measuring from 0 to 40 mm of water;

(b) filled with water and able to measure from 0 to 160 mm of water.

Recording quantity measuring instruments are built to measure pressure drops up to 100 mm of water.

3. Micromanometers. The water gauges described above have one serious disadvantage; the possible accuracy of reading does not

exceed 0 5 mm of water and they are therefore unsuitable for measuring low pressures of an order of 2-3 mm, in particular for research work, when great accuracy is needed. For such work micromanometers are used

One of the most widely used is the inclined micromanometer. Its principle can easily be seen from Fig 5-5 The legs of the U-tube are made of different diameters (S_1 $S_2 = 700$) so that during the measurement of a pressure, the liquid shall move only in the narrow leg and the level of the wide leg remains practically constant, enabling readings to be taken only in the narrow leg. The narrow leg is sloping so as to multiply the movement of the meniscus. If for a certain pressure difference the movement of the meniscus in a vertical tube is aa_1, then in the inclined tube, the movement bb_1 of the meniscus for the same will be

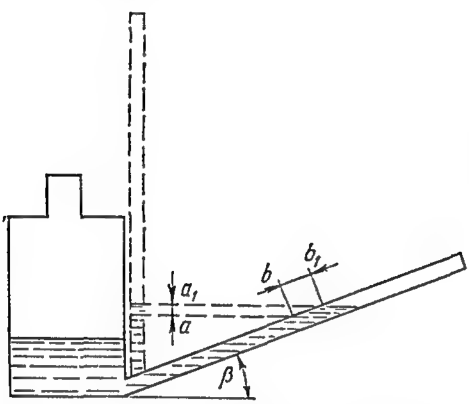

Fig 5-5 A micromanometer, schematic

$$bb_1 = aa_1 \quad \sin\beta$$

where β = angle of inclination of the tube

From this equation we obtain

$$aa_1 = bb_1 \sin\beta$$

The factor $\sin\beta$ is usually denoted by the letter F and is called the *slope factor* The angle of slope is chosen such that the value of the slope factor F is a simple fraction, for example 0 5, 0 25 or 0 125.

The liquid used is generally alcohol If the specific gravity of the alcohol is Δ and the reading on the inclined scale is h_{incl}, then the value of the elevation in vertical water gauge will be

$$h_{incl} F \Delta \xi \tag{5-2}$$

where ξ is the correction factor of the micromanometer

Micromanometers should be provided with a calibration certificate

The type MMH micromanometer is shown in Fig 5-6 It is designed for measuring pressure drops within the range of 0-200 mm water.

On the silicon-aluminium alloy base *1* is fixed a steel sheet tank *2* with a sealed lid above and a rubber gasket On the lid are fixed a three-way cock *3*, a filling plug *4*, and an adjuster *5* for the zero position of the meniscus, which regulates the position of the column of alcohol in the measuring tube *6* to the zero division of the scale The measuring tube is placed on a board hinged to the base *1*, the lower part of the tube is connected to the tank *2*

through a fitting and rubber tubing, and its upper part is joined to the three-way cock *3* through another rubber tubing The measuring tube *6* is installed in such a way that the zero of the scale is at the axis of rotation of the board. The scale of the measuring tube is 250 mm long and each division is 1 mm.

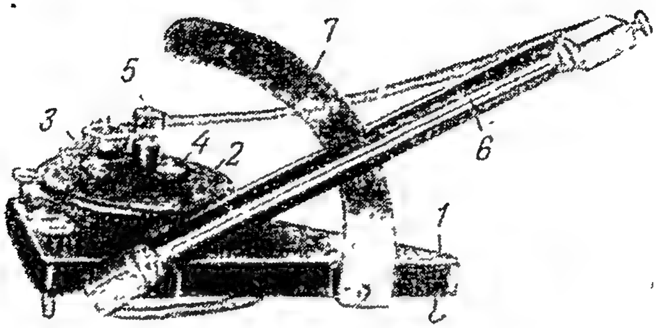

Fig 5-6 Type MMH micromanometer

To set the board and measuring tube at the required angle of inclination, an arc *7* is fixed to the base *1*, having five openings marked with the figures 0.8, 0 6, 0.4, 0.3, and 0.2 indicating the slope factors of the instrument.

The micromanometer must be installed in a strictly horizontal position and the base *1* therefore has two levels with cylindrical bubbles

The instrument is filled with alcohol through the opening in the lid; the alcohol is removed through a drain cock in the lower part of the tank The ducts in the three-way cock are placed so that when the cock is turned anti-clockwise as far as it will go, the tank and measuring tube are connected to the atmosphere, while the openings at the fittings *1* and *3* (Fig 5-7) are shut off, in this position of the cock the position of the meniscus is checked to the zero of the scale. As the cock is turned in a clockwise direction to the stop, the fitting *3* is connected with the tank, the fitting *2* being connected with the fitting *1*, and through it to the glass measuring tube; as this is done the opening to the atmosphere is closed.

When a pressure drop is being measured by the instrument, the rubber tubing leading from the point of measurement is connected to the fitting *2*; but when the pressure difference is measured, the tube leading from the point with the highest pressure is connected to

the fitting *3*, and that from the place with the lowest pressure is connected to the fitting *2*; during measurement the alcohol will rise in the tube and fall in the tank. The MMH micromanometer is operated in the following way.

1 The instrument is filled with alcohol To do this, the instrument is placed in a horizontal position, the filling plug is unscrewed, and the tank is filled with ethyl alcohol having a density of 0.8095 g/cm³ until its level in the measuring tube is more or less opposite the zero of the scale; the plug is then screwed in again

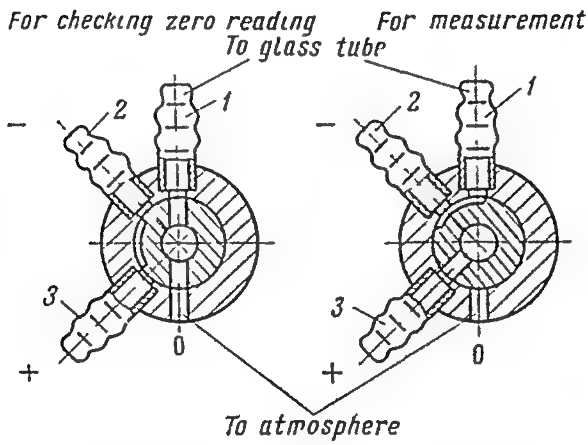

Fig 5-7 Positions of the three-way cock in the type MMH micromanometer

until it is tight; the board and measuring tube must then be slewed to the highest position, that is, opposite the opening on the arc *7*, marked with the figure 0 8 (Fig. 5-6).

2. A check must be made to see that there are no air bubbles; to do this, on the fitting *2* (Fig. 5-7) of the three-way cock a small length of rubber tubing is placed and the cock is set; the level of the alcohol in the measuring tube is raised by sucking up, roughly to the level of the end of the scale; no air bubbles should be seen in the column of alcohol when this is done; if there are some air bubbles they must be removed by blowing the alcohol back into the tank Large quantities of air heated by the lungs must not be blown into the tank.

The instrument is now ready for work It is set horizontally at the place where the pressure drop is to be measured, the best way of setting the instrument underground is on the instrument box which must first be set firmly on the floor, the angle of slope of the measuring tube then being fixed at 0 8 for the first measurement.

The meniscus is then finally set at zero, for which the three-way cock must be rotated anti-clockwise up to the stop, and using the adjusting drum, the meniscus is moved to the zero position.

The micromanometer is connected to the points between which the pressure drop has to be measured, the three-way cock is turned clockwise as far as it will go and the gauge tube is read.

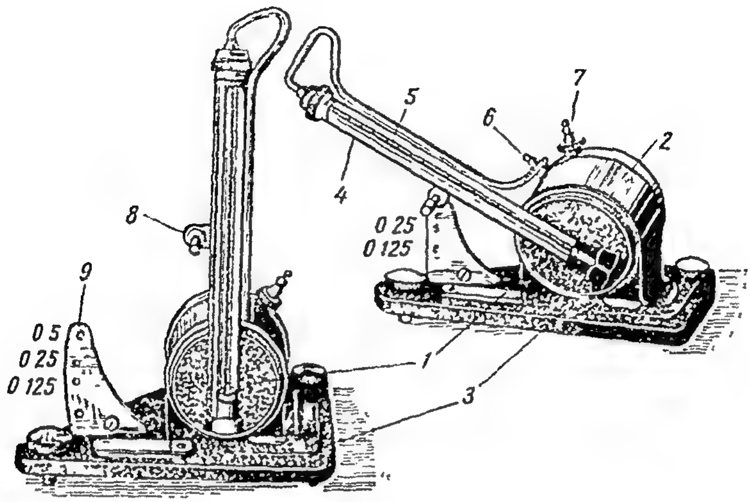

Fig. 5-8 The ЦАГИ micromanometer

To make the meniscus easily readable, the alcohol should be tinted with methyl red at a dilution of 0.05 g/litre.

In the MMH instrument the ratio between the cross-sectional areas of the tank and the gauge tube is 485.

The measured pressure drop in mm of the sloping column of alcohol is converted to mm of vertical water gauge by the formula

$$h = KCh_{incl} \qquad (5\text{-}3)$$

in which K = instrument constant, equal to $\sin\alpha\,(1 + 0.0279)$;
where α = angle of inclination of gauge tube
C = correction factor of the micromanometer
h_{incl} = manometer reading, mm

If the density of the alcohol is not 0.8095 g/cm³, the effects of temperature and of strength of the alcohol are adjusted by the correction factor C provided in a table supplied with the instrument.

A disadvantage of this micromanometer is the hinged joint between the gauge tube and the tank, which may cause bubbles. The ЦАГИ micromanometer does not have this disadvantage.

The ЦАГИ inclined micromanometer (Fig 5-8) consists of the following main parts, a closed cylinder *1* freely rotating within

a clamp *2* fixed to the cast-iron base *3* of the instrument. Firmly fixed to the cylinder is an open clamp *4* with a glass scale tube *5* set in it; the tube *5* is connected at one end to the cylinder *1*, and at the other end to a narrow bent tube soldered to the cylinder *1* and supplied with an adapter *6* for connecting rubber tubing to it, a similar connection *7* is provided into the hollow cylinder *1*. The clamp *4* has a pin *8*, by which it can be fixed on the segment *9* with openings for the pin, each opening corresponding to a different slope factor 0 5, 0 25 and 0.125. The base *3* has two adjusting screws and two bubble tubes perpendicular to each other

A pressure is measured in the following way·

(1) The instrument is filled with alcohol* until it appears in the bottom end of the glass tube and rises 5-10 mm in it.

(2) The instrument is set horizontally using the alcohol levels

(3) The tube *4* is set at the chosen slope

(4) The zero reading is taken and recorded with the micromanometer open

(5) The adapter *6* is connected by rubber tubing to the point, e g the fan drift, whose pressure is required**.

(6) The reading is noted***, that is, the scale division at which the alcohol meniscus stops

(7) From the final reading, the zero reading is subtracted, and the difference obtained is converted to mm of vertical water gauge by the equation

$$h = h_{incl}\ F\Delta\xi \text{ mm water} \tag{5-4}$$

To determine the density of the alcohol, the air temperature must be measured and the density Δ must be determined from the calibration curve

The ЦАГИ micromanometer is an accurate, reliable instrument which has been fully proved in laboratory and underground work.

4. Pitot tubes. To transmit the pressure to the water gauge, use is made of rubber tubes 5-10 mm in diameter with Pitot tubes installed at the points of measurement of the pressure differences It is explained in hydraulics that Pitot tubes are built in such a way that they can measure at will the static head h_{st}, the velocity head h_v; and the total head h_t equal to the algebraic sum of the velocity head and the static head

---

* The density of the alcohol at different temperatures should be determined beforehand

** This is for the measurement of a pressure drop, for the measurement of a pressure head, greater than atmospheric, the adapter *7* is used

*** If the meniscus fluctuates, several readings must be taken, every ten seconds or so, or a number of consecutive readings must be recorded close to the average position of the meniscus, the extreme readings are ignored

Figure 5-9 shows a Pitot tube with a hemispherical head. As the drawing shows, the tube consists of an endpiece *1*, a holder *2*, and two connections *3* and *3'* to which rubber tubes are fitted for connec-

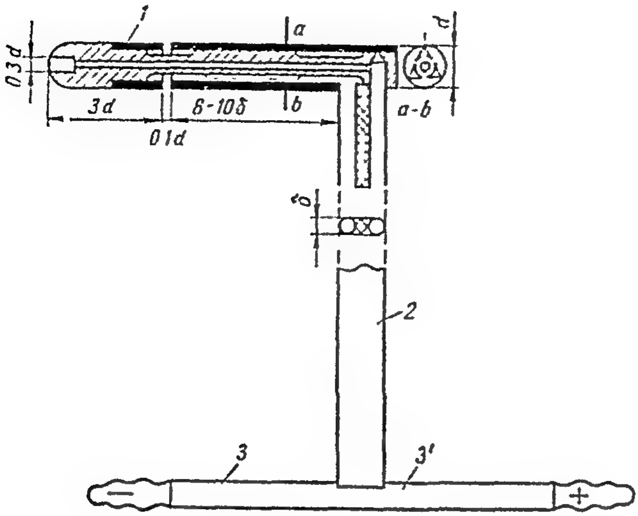

Fig. 5-9 Pitot tube

tion to the water gauge. The endpiece has a central opening, connected through the holder to the tube *3'*, marked by the plus sign, and a ring-shaped slot connected to the other tube *3*, marked with the minus sign. During a pressure measurement, the open end of the endpiece faces the air flow.

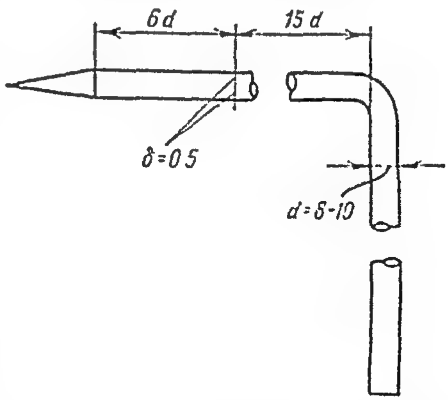

Fig 5-10. Static tube for measuring only static pressure or head

Now depending on which end of the Pitot connections *3* and *3* should be joined to the water gauge (one leg of which is connected to the atmosphere) the water gauge can measure either the total head, or the static head or the velocity head If the "minus" end *3* is connected, the reading is the difference between the atmospheric pressure and the actual pressure at the point of installation of the tube, that is, the static head h_{st} If the connection *3* marked by the plus sign is joined to the water gauge, the total head is measured, that is, the algebraic sum of the static head h_{st} and the velocity head h_v; that is to say if a suction is being measured, the value is $h_{st} - h_v$ and if a positive pressure

is being measured the value is $h_{st}+h_v$; if the "plus" and the "minus" ends are joined, the measurement indicated will be the velocity head h_v at the endpiece of the Pitot tube.

To measure only the value of h_{st}, that is, the static head, it is possible to use the so-called static tube (Fig. 5-10), that is, a tube with an endpiece which has only a side slit or some small openings of a diameter not exceeding 0 5 mm, but no central hole.

The Pitot tubes made in the USSR are usually 10 mm in diameter.

5-5. MEASUREMENT OF AIR VELOCITY

The velocity of air flow is measured daily by the ventilation staff.

Supervision of the correct air distribution underground, the determination of the air losses on the way to the face, the calculation of the quantity of gases formed underground and a number of other investigations would be impossible without accurate measurement of the air

5-5 1 Average and Maximum Velocities of Air Flow

The velocity of air flow underground varies widely, from a few centimetres to 10-15 m/sec or more At very low velocities (a few centimetres per second) the air seeps through falls, worked-out areas, packs, and cracks in pillars

Low air velocities (from 0 05 to 0 2-0 3 m/sec) are often observed in the stopes of metal mines or salt mines, and generally in badly ventilated places Average velocities of 1-5 m/sec are found in shafts, main roads, cross-measure drifts, etc. Higher velocities exceeding 10 m/sec usually occur only in the fan drifts and at the outlets of fan casings with no evase

The average velocity of air flow v_{av} in a mine airway must be clearly distinguished from the varying velocity v at any point in it The average velocity is found by moving the air measuring instrument throughout the cross section of the roadway. Multiplied by the cross section of the roadway S the average air velocity gives the total quantity of air

$$v_{av}\cdot S=Q$$

Conversely, if the air quantity Q is known the average velocity is $\frac{Q}{S}$

The velocity varies throughout the cross section Usually the maximum velocity v_{max} is in the centre of the airway.

Usually when the velocity of air flow is mentioned, the average velocity is meant; this is the velocity which is considered in the application of the Safety Regulations for the maximum or minimum air

velocity. The average velocity, as explained, also gives the total air quantity passing through the airway. The maximum velocity is only rarely of interest, but it does control the raising of dust, or the blowing of the flame of a safety lamp through the gauze, etc.

The value of the ratio $v_{av} : v_{max} = \psi$ depends on the roughness of the airway. V. N. Voronin has developed the following equation for determining ψ:

$$\psi = \frac{1}{(1+11.8\sqrt{\alpha})} \tag{5-5}$$

where α = friction factor (see Chapter 8); for example, when $\alpha = 0.0010$, $\psi = 0.72$; with $\alpha = 0.0015$, $\psi = 0.68$; and when $\alpha = 0.0020$, $\psi = 0.64$. From experimental data these values seem low; on the average, one can take $\psi = 0.75$ to 0.80. The value of ψ can be assumed with enough accuracy for practical purposes to be independent of the average air velocity (see page 244).

5-5.2 *Techniques of Measuring Air Velocity*

Various techniques of measurement are used depending on the purpose of the measurement and on the value of the air velocity.

Measurement of average velocity. 1. The average velocity can be determined in relation to the air velocity at the centre of the airway by the equation

$$v = \psi v_{av} \tag{5-6}$$

The method is inaccurate and is only used when, because of the low air velocity, an instrument such as an anemometer is not traversed through all points of the cross section.

2 The so-called method of areas in which the cross-sectional area of the roadway is broken down into imaginary small areas $s_1, s_2, s_3, \ldots \ldots s_n$ (25 or more); in the centre of each small area the air flow velocity is measured ($v_1, v_2, v_3, \ldots \ldots v_n$), after which the average velocity is determined from the formula

$$v_{av} = \frac{s_1 v_1 + s_2 v_2 + s_3 v_3 + \ldots + s_n v_n}{S} \text{ m/sec} \tag{5-7}$$

This method is accurate but tedious, requiring much time, and is used in scientific investigations Simultaneously with the measurement of the velocity v_1, etc., the air velocity v_c must also be measured at a control point in the airway at the measuring station. After all the velocities have been measured in the small areas the value of the average control velocity $v_{av\,c}$ is determined and all the measured air velocities are then reduced to this average velocity by a factor $\frac{v_{av\,c}}{v_c}$,

In the cup-type anemometer the moving air strikes four hemispherical cups, mounted on the ends of two mutually perpendicular rods fixed on a shaft, forcing it to rotate; the rotations are noted by a counter which indicates the rotations or a number proportional to the rotations of the anemometer shaft. The counter has a large needle which

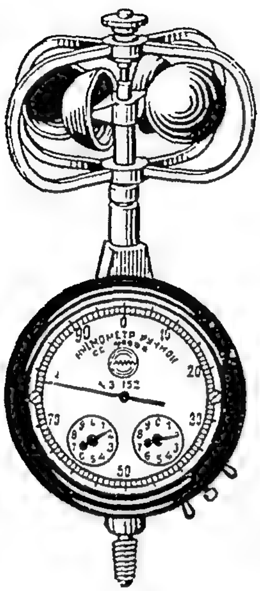

Fig 5-12 Cup-type anemometer

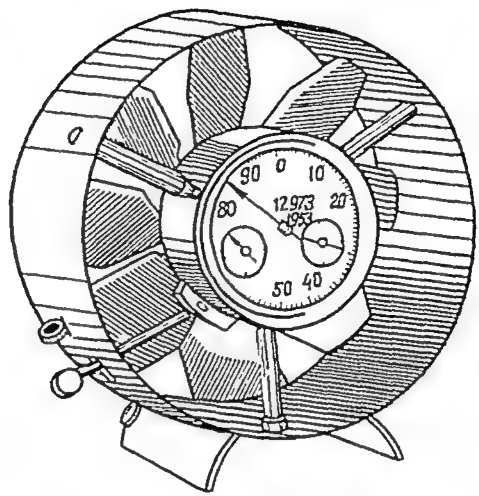

Fig 5-13. Vane anemometer

indicates units and tens, and smaller needles which indicate hundreds and thousands of rotations; the anemometer is started and stopped by a small lever on its body

In the vane anemometer, as its name implies, the moving air strikes a number of plates or vanes (usually eight) fixed to thin rods on a shaft; in so-called engineering anemometers, the shaft is held in two bearings; in other anemometers (type ACO-3), the shaft is a thin stem carrying the vanes (Fig 5-14)

The air velocity is measured by the anemometer in the following sequence. The readings of all the needles are taken, beginning with the largest, since this ensures a correct record if the indications of the other dials are not entirely clear. The air measurement is made and the final readings are recorded, the first readings are subtracted from the final readings and the difference is divided by the measurement time in seconds; the quotient, which is the number of scale divisions per second, is converted to m/sec by the calibration chart which accompanies the anemometer.

This calibration certificate or chart is drawn up by various institutions in various forms The most widely used certificate is in the form of a calibration curve or a straight line plotted on graph paper with ordinates as the number of divisions per second, sometimes per minute, against a horizontal scale of velocity in m/sec, sometimes m/min (Fig 5-15).

If the curve *ab* is prolonged to enter the area of low velocities, it will intersect the abscissa axis at a point *c*; the length *ac* of the curve

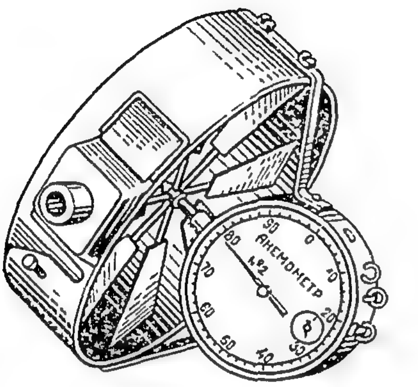

Fig 5-14 Vane anemometer on a stiff wire spindle, used for measuring low air velocities, type ACO-3

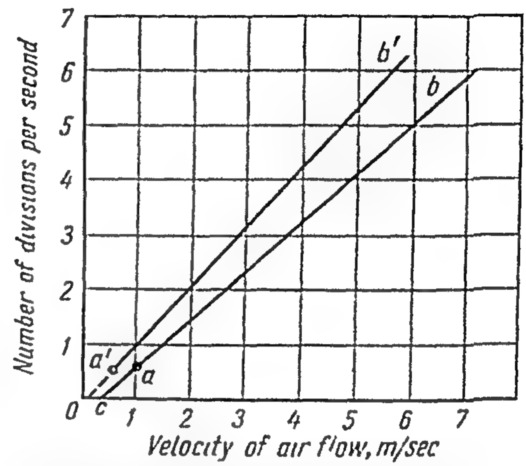

Fig 5-15. Calibration chart for anemometer, in the form of a straight line

will be little reliable but it can be used for measurements which do not require high accuracy, such as measurements of mine air velocity However a calibration curve which when prolonged intersects the ordinate axis is much less reliable, in fact it should intersect the axis of abscissas (the line *a'b'*).

Calibration curves can also be presented in the form of an equation: $v = a + bn$, or $v = bn$, in which a is the abscissa intercept between the curve and the origin, b is the cotangent of the slope of the curve and n is the number of scale divisions per second For low velocities the equation is more accurate than the graph

Calibration certificates can also be drawn up in the form of a table (Table 5-1).

In this instance the velocity is equal to the number of scale divisions, multiplied by the factor k

If necessary, anemometers can also be calibrated at the mine; a small bench is built up, consisting of a base, holding a vertical crankshaft with the crank above it, at one end of the crank the anemom-

TABLE 5-1 **Calibration Certificate**

Number of divisions per second	Multiplying factor, k	Number of divisions per second	Velocity, m/sec
0.5	1 36	2 0	2 1
1 0	1 1	3 0	3 1
etc		etc	

eter is mounted at a radius r from the centre of the shaft, the anemometer being rotated by the shaft, either by hand or by a small motor. If n is the number of turns of the shaft in t seconds, the true rotational speed of the anemometer will be equal to:

$$v = \frac{2\pi rn}{t} \text{ m/sec}$$

With the values of v and n found for several rotational speeds, it is not difficult to draw up a graph or a table for the factor k; $k = \frac{v}{n}$, in which n is the number of scale divisions in the time t.

If the atmosphere in which the anemometer works is dirty or saturated with water vapour and the anemometer is in daily use, it will need to be calibrated frequently, e.g. in underground conditions every two to three months; if the air is clean, the anemometer can work 6-12 months without calibration.

If the anemometer readings are not reliable and the mine has a correctly calibrated anemometer, it is possible to check the faulty anemometer in the following way. two, preferably three, airways in the mine are chosen, having different air velocities, for example about 1-2 m/sec or 4-5 m/sec or more; in each airway, a measuring point is chosen on a straight length; at these measuring points the air velocity is measured three times with both instruments, using alternately the correct instrument and the one being checked; the arithmetic means of v_i and n_i are calculated; from these values a graph or a table of corrections can be drawn up.

In making anemometer measurements the following must be remembered

(1) The measurement must be made in a straight length with smooth supports, close to the walls of the roadway, and with a clean floor. Measurements must not be made behind mine cars or near curves in the roadway, preferably at least 5 diameters from a turn.

(2) Not less than two, preferably three, measurements should be made; the deviation between individual measurements should not be more than 2-3 per cent.

(3) When a measuring station is used, the measurement should be made at 0 5 m from the downstream end of the station.

Below are given some interesting results of British investigations into the use of anemometers in mines

(a) the skew of the anemometer in relation to the air flow; with a skew of 20° the error is not more than 2 5 per cent,

(b) movement of the anemometer in the cross section; the greater the speed of traversing v and the less the air velocity v_1, the greater is the error, e g at a traversing speed $v = 9$ m/min and an air velocity v_1 of 60 m/min, the error is 1 1 per cent; at $v = 18$ m/min it is equal to 4 4 per cent and with $v = 36$ m/min it is 15 6 per cent;

(c) slow starting of the anemometer because of its inertia; this effect is not noticeable when the measurement lasts at least 60 sec, preferably 100-120 sec,

(d) alternating position of the anemometer in zones with different air velocities, this effect cannot be eliminated;

(e) inopportune giving of the starting signal and non-repeatability of the traverse of the anemometer;

(f) effect of the operator's body, the anemometer should be tied to a stick 1.5 m long, or longer in high roadways, if this is done the correction factor to the measurement is 1 0,

(g) in some anemometers (without handles), holding them in the hand can increase the air velocity by 11-12 per cent,

(h) change of specific weight of the air, according to Ower the correction is $\sqrt{\frac{\gamma_1}{\gamma_0}}$, where γ_1 and γ_0 are the specific weights of air at calibration and when the anemometer is used.

A small difference may occur when the anemometer or the operator is changed (not more than 2 per cent); in far-reaching investigations a personal correction factor should be applied for each operator.

If the measurement is correctly made, the error by this method, compared with the method of areas, does not exceed 2 5 per cent.

5-5 4 *Measurement of Low Air Velocities*

Measurement with Specially Sensitive Anemometers

(a) Using a large anemometer, of 18 cm diameter with vanes made of mica

(b) In the swinging-plate anemometer (Fig 5-16) a swinging plate *3* is moved by the air stream through connection *2* The plate swings on a shaft in the plane-cylindrical box *1* and is linked to a needle travelling over a scale on the transparent cover of the instrument The air velocity scale is graduated in tenths of a metre per second

A magnetic damper ensures smooth operation The plate is returned to its zero position by a small spring anchored at the hinge. The

hollow space *4*, in which the plate swings, gradually widens beside the plate, thus enabling a wide range of velocities to be measured, since part of the air passes beside the plate The instrument can measure velocities from rather less than 0.1 m/sec up to 10 m/sec.

(c) Using a ventilated (differential) anemometer working with a small fan (Fig. 5-17); the fan *1*, which is driven by a spring, forces air through the vertical slot in the tube *2*, turning the vaned wheel

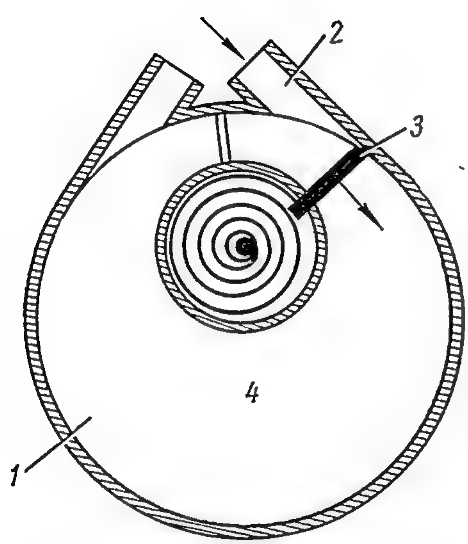

Fig. 5-16 The ДЕСУ swinging-plate anemometer (diagram)

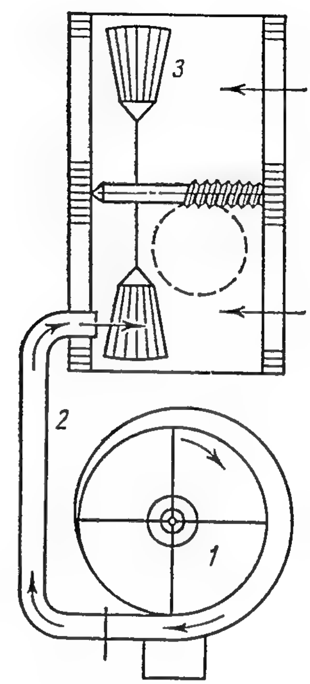

Fig 5-17 Diagram of a differential anemometer with a clockwork fan

3 at a known rotational speed in still air. In an air flow (direction shown by arrows) the wheel is either slowed down, or accelerated. Usually it is held so as to be slowed down. If this is so the measured air velocity is equal to the difference between the anemometer speed in still air and that at the measuring point

The spring which drives the fan *1* is wound with a key To obtain accurate readings it is important that the number of turns of the wheel shall be constant Therefore one factory recommends that not more than one or two measurements shall be made before winding, otherwise the fan speed drops.

5-5 5 *Measurement of Air Velocity in Scientific Investigations*

Apart from the usual measurements special methods are also used. These include mainly the subdivision of the measured cross section into small areas (page 215):

(a) measurement with the Kata thermometer (Part 1),

(b) measurement with Pitot tubes. From the equation

$$h_v = \frac{v^2}{2g}\gamma$$

we have

$$v = \sqrt{h_v \frac{2g}{\gamma}} \tag{5-8}$$

If we call F the slope factor of the micromanometer, and Δ the specific gravity or density of the alcohol, and ξ the correction factor of the micromanometer, then

$$v = \sqrt{h_v \frac{2g}{\gamma} F\Delta\xi} \text{ m/sec} \tag{5-9}$$

in which h_v = length of the sloping column of alcohol

γ = specific gravity of air at the given temperature and pressure;

(c) hot-wire anemometers; in these instruments, the air velocity is measured by the change in resistance of a hot wire forming one of the arms of a Wheatstone's bridge, the change depending on its change in temperature at different air velocities In the hot-wire anemometer of K N Vasiliev, the temperature of the hot wire is directly measured by a thermocouple.

The instrument designed by E. A. Batmanovsky is also based on the Wheatstone's bridge and is designed for mine air measurements Tested in laboratories and in a non-ferrous metal mine it was found to have a reading error not exceeding 2 per cent; its sensitivity is 5 cm/sec

A new method of measuring air velocity involves the use of radioactive isotopes In this method a gas, for example an isotope of methyl bromide, is released into the air flow, and as the molecules pass a special counter they are recorded The distance at which the gas becomes completely mixed with air, varies from 30-100 D in a smooth airway down to 10 D in a rough airway, D being the diameter of the airway.

5-5 6 *Calculation of the Air Quantities Flowing Underground*

To determine the quantity of air Q flowing through any roadway, the average velocity, determined by one of the methods mentioned above, should be multiplied by the airway cross section S. The air

quantity is calculated in m³/sec or m³/min to the nearest 10 m³/min when Q is less than 500 m³/min and to the nearest 25 m³/min when Q is more than 500 m³/min.

The area is measured by a tape with an accuracy of 1 cm In timbered airways the height is measured inside the frame, the width being the average width In irregular airways without supports, a prop is set in the centre of the roadway and offsets are taken every 20-25 cm from it to the walls, from these dimensions the cross section is drawn, and the area is calculated by planimeter If no planimeter is available, the cross section is drawn out on squared paper, and its area is found by counting the number of squares in the area

The accuracy of determination of the air quantity using the anemometer depends on the accuracy of the measurement of the air flow and on the airway cross section. If there are experimental errors, it is known that the total error is equal to the square root of the sum of the squares of the errors. On the average it is possible to state that in careful mine measurements of air quantity the error can be limited to 3-5 per cent.

5-5.7 Special Instances of Determination of Air Quantity

1. *The measurement of the air quantity flowing through the fan.* We shall use one of the following methods.

(a) measurement in the fan drift by the method "in front of the operator" or "in the section";

(b) measurement at the outlet of the evase (used when there is a more or less uniform distribution of air velocity across the evase outlet),

(c) in axial-flow fans the air is measured at the outlet from the ring-shaped space of the fan, occupied by the fan blades; the measurements are made at 8 points, on four mutually perpendicular diameters, and their mean is calculated.

The air quantity at the outlet from the evase of a fan is measured by the "point" (or area) method. The plan of the evase (Fig 5-18) is first drawn to scale on paper and at distances of 10-15 cm from the edges, lines are drawn to locate the extreme points, after which the distance between these extreme points is divided into intercepts 40-50 cm long and through the points a, a, . . . b, b, . . . mutually perpendicular lines are drawn The intersections of these lines will be the points of measurement of air velocity. To fix them in the actual evase, chalk lines are drawn on the walls to indicate a, a, b, b, etc.

The anemometer is fixed on a stick about 1 m longer than needed to cover the evase, so that it can be held in the hands.

For convenience in measurement, the rod should first be marked with chalk.

The average velocity is obtained in the following way. the first reading of the anemometer is recorded; the anemometer is placed in the appropriate point at one of the corners and is started; after 30 seconds the anemometer is moved to the next point, without reading it, held there for 30 seconds, moved on to point *3* and so on until all the points have been covered The chalk marks on the walls of the evase and on the rod are used to locate the anemometer correctly at the

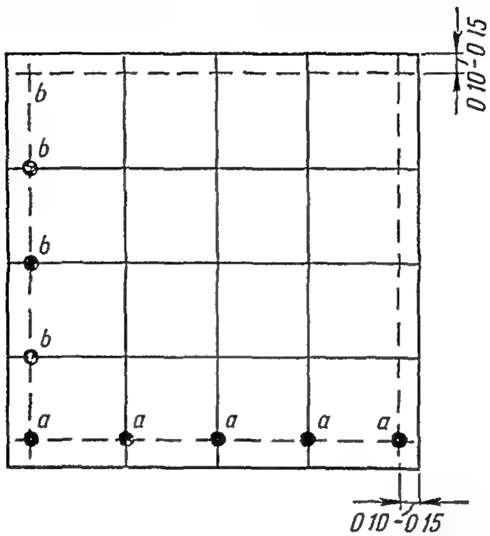

Fig 5-18 Measurement of the air flow at the outlet from the evase, using the point method

points After the traverse has been completed the final reading is noted and the original reading is subtracted from it, the difference is divided by $30n$, where n is the number of points; the quotient is evidently equal to the number of scale divisions of the anemometer per second

After the first traverse, a second traverse should be made for checking, and the arithmetic mean of the number of divisions per second determined The difference between the two measurements should not exceed 2-3 per cent The measurement is made by two persons, one person holds the anemometer rod in his hand moving it from one point to another. The second person stands close by with the stopwatch in his hand and signals to the first person to move the anemometer every 30 seconds Measurements at the evase are reliable only when the air stream does not break away from the walls of the evase

If simultaneously with the traversing of the cross section of the evase, a second anemometer is used to measure the air velocity v' at any point of the cross section it becomes possible to calculate the

so-called *discharge coefficient m* equal to $\frac{Q}{v'}$; this coefficient for any measured air velocity v' at the given point enables the discharge to be calculated without repeating the traverse

$$Q = v'm \text{ m}^3/\text{sec} \tag{5-10}$$

Equation (5-10) can be used only in evases with steady air flow

2. If an integrating flow meter is installed for measuring the air quantity, its readings must be checked by determining the fan output by one of the methods listed above; the need for this check arises from the fact that the integrating flow meter may be calibrated in conditions different from those at the mine.

3. When the air velocity is measured in a vertical shaft, it must be remembered that anemometers are generally calibrated in a horizontal air flow, and therefore their readings may be not fully reliable; in small pits with ladder compartments the work is done as follows from the ladder compartment the air velocity is measured in the centre of the ventilation compartment and multiplied by a correction factor of 0.75-0.80 for close timbering; the air velocity is also measured in the opening of the ladder platform; the two air quantities are then calculated and added together

4. In measuring the average air velocity in a regulator or small opening, the anemometer must be held in the hand and traversed throughout the section of the opening, the total number of anemometer scale divisions being multiplied by 0.90 in small openings and by 0.95 in large openings.

In measuring the average air velocity through a door, the observer must stand at the side of the open door, without obstructing it,

TABLE 5-2 **Correction Factors for Anemometers**

Measurement at outlet from ducting				Measurement at inlet to ducting			
Ducting diameter, mm	Air velocity, m/sec	Correction factor		Ducting diameter, mm	Air velocity, m/sec	Correction factor	
		Type A	Type B			Type A	Type B
360	2.5-12.5	0.84	0.84	360	8-12.5	0.78	0.79
540	2.5-20	0.85	0.90	540	8-12.5	0.82	0.82
720	4.0-23	0.85	0.89	540	17-23	0.79	0.82
900	2.5-13.5	0.86	0.86	720	7.5-13.5	0.80	0.81
				720	14.5-22.5	0.79	0.82

then holding the anemometer in his outstretched hand or on a stick, he traverses it over the cross section; the correction factor for fairly large doors is nearly unity If S_{door} S_{airway} is less than or equal to 0 5, then $k = 0\ 95$

5. Measurement of the air quantity at the inlet to and the outlet from ventilation ducting. To obtain reliable results underground it is recommended that

(a) the measurement should be made by the anemometer in the centre of the ducting; the anemometer is fixed to a short stick, which is pressed against the flange of the ducting during the measurement;

(b) the fan should be not nearer than 25 D (D is the diameter of the ducting) from the measuring point,

(c) to obtain the average air velocity v_{av}, the speed in the centre is multiplied by a correction factor k, these factors are given below for anemometers of two types with a dial in the centre of the vanes (type A), and with a dial at the side of the vanes (type B, the Casella anemometer)

CHAPTER 6

AEROSTATICS

6-1. DERIVATION OF FORMULAS FOR CALCULATING THE INCREASE OF PRESSURE WITH DEPTH

These formulas are required for calculating the natural ventilating pressure (Chapter 10) which is equal to the difference between the two pressures p_1 and p_2 in the downcast and upcast shafts.

In hydraulics the following equation of equilibrium for heavy fluids is developed:

$$dz = \frac{dp}{\gamma}, \quad \text{or} \quad dp = \gamma\, dz \tag{6-1}$$

where dz = increase of height of the column of liquid. If the specific weight or density is constant, which in practice is true for liquids, the pressure can be calculated from the following simple equation

$$p = p_0 + \gamma\, dz \tag{6-2}$$

in which p_0 = pressure at the top of the column of liquid.

This equation can be used also for calculating the pressure of a column of air, subject to the condition that the density of the air is either constant or assumed constant or equal to the arithmetic mean of the densities at the top and the bottom of the column of air.

When the specific weight is variable, Equation (6-1) after integration becomes

$$p = p_0 + \int_{z_0}^{z} \gamma\, dz \tag{6-3}$$

in which γ = density changing with depth.

As mentioned above, the change in the state of the air underground is irregular, and therefore the specific weight also changes irregularly. Since to arrive at a solution of Equation (6-3) it is necessary to start from some regular change of density without which the solution is impossible, we will consider the following known simple changes in the state of the air:

change at constant volume, v = constant, γ = constant
change along an isotherm, T = constant
adiabatic change, $dQ = 0$
polytropic change, pv^n = constant

So as to decide which of these laws comes nearest to the actual and much more complex change in the air state in mine shafts we shall proceed as follows

(a) we shall deduce for each of these laws an equation for calculating the increase of pressure with depth,

(b) we shall find by how many degrees the air temperature should change per 100 m, for the given law to be valid;

(c) we shall compare the calculated results of the change of air temperature with the actual results observed in mine shafts, this comparison will enable the most fitting equation to be chosen.

Isochoric Change. So that the specific volume of the air shall be constant or, which comes to the same thing, the density γ shall be constant as the air passes down the shaft, the increase in density due to the increased pressure must be compensated by a reduction in density due to an increase in temperature

From the conditions of equality of densities at the surface γ_0 and at a depth of 100 m γ_{100}, we have

$$\gamma_0 = 1\ 2 \times \frac{p_0}{760} \times \frac{288}{T_0} = \gamma_{100} = 1\ 2\,\frac{p_{100}}{760} \times \frac{288}{T_{100}}$$

or

$$T_{100} = T_0\,\frac{p_{100}}{p_0} \tag{6-4}$$

where T_0 and p_0 = temperature and pressure of the air at the surface

T_{100} and p_{100} = temperature and pressure of the air at a depth of 100 m

Assuming the increase of pressure per 100 m to be equal on the average to 0 9 mm of mercury, which corresponds to the pressure interval 760 to 780 mm of mercury at 15°C, we obtain from Equation (6-4) assuming $T_0 = 288°$ C and $p = 760$ mm

$$T_{100} = 288\,\frac{(760 + 9\ 4)}{760} = 291.56°$$

that is, for the density to be constant with increasing depth the temperature must increase every 100 m by $\Delta t = 291\ 56 - 288 = 3.56$°C. The general solution is $\Delta t = 3\ 42$°C These data refer to dry air

Isothermal Change. From Clapeyron's equation, $pV = RT$; by substituting $\frac{1}{\gamma}$ for V, we obtain $\frac{1}{\gamma} = \frac{RT}{p}$, substituting this expression in the equation $dz = \frac{dp}{\gamma}$ and integrating between the limits p_0 and p_1, z_0 and z, calculating along the z-axis, directed vertically downwards, we obtain

$$\int_{p_0}^{p} \frac{dp}{\gamma} = \int_{p_0}^{p} \frac{RT}{p}\,dp = RT \log_e \frac{p}{p_0} = \int_{z_0}^{z} dz = H$$

or

$$\log_e \frac{p}{p_0} = \frac{H}{RT}$$

Converting this equation from natural logarithms to logarithms to the base 10 and substituting 29 27 for R we obtain

$$\log \frac{p}{p_0} = \frac{H}{2\ 3 \times 29\ 27 \times T} = 0.0148 \frac{H}{T} \approx 0\ 015 \frac{H}{T}$$

from which

$$\log p = \log p_0 + 0.015 \frac{H}{T} \tag{6-5}$$

where T = air temperature in the shaft (or average temperature if T_0 is not equal to T_H).

Adiabatic Change. When no heat is either taken from the air nor added to it, the temperature of the air as it passes down increases by about 1°C (precisely 0.98°) for every 100 metres of shaft; evaporation of water or condensation of water vapour will reduce or increase this temperature. In free air above the earth's surface, the temperature gradient, or change of temperature per 100 m, is close to 1°C in summer; in October the gradient on the outskirts of Leningrad diminishes to 0 55°C, and in December it falls to 0 03°C, in January and February the gradient becomes negative, from 0 24 to 0.17°C per 100 m

Polytropic Change. Let us first note the assumption that the change of temperature with depth is linear

$$t_H = t_0 + aH$$

or

$$T_H = T_0 + aH \tag{6-6}$$

where t_H and t_0 = temperatures of air at depth H from the surface, and at the surface, respectively

a = increase of temperature per metre depth

We will start at first from Equation (6-1), and substituting in it $\frac{RT}{p}$ for $\frac{1}{\gamma}$ and H for z, we obtain

$$\frac{dp}{p} = \frac{dH}{RT}$$

or, taking account of Equation (6-6)

$$\frac{dp}{p} = \frac{dH}{R\,(T_0 + aH)}$$

Integrating this equation between the limits of p_0 and p, and from 0 to H we obtain

$$\log_e \frac{p}{p_0} = \frac{1}{Ra} \log_e \frac{(T_0 + aH)}{T_0}$$

or

$$\log \frac{p}{p_0} = \frac{1}{Ra} \log \frac{T_H}{T_0}$$

but from Equation (6-6) $a = \frac{T_H - T_0}{H}$, consequently, we obtain

$$\log \frac{p}{p_0} = \frac{H}{R(T_H - T_0)} \log \frac{T_H}{T_0}$$

or

$$\log p = \log p_0 + \frac{H}{R(T_H - T_0)} \log \frac{T_H}{T_0} \tag{6-7}$$

Example. Given $H = 500\ m$, $t_0 = 5°C$, $t_H = 15°C$, $p_0 = 760$ mm Determine p

Solution. According to Equation (6-5)

$$\log p = \log 760 + 0\ 015 \times \frac{500}{283} = 2\ 90732 \quad \text{and} \quad p = 807\ 82;$$

from Equation (6-7)

$$\log p = \log 760 + \frac{500}{29\ 27 \times 10} \times \frac{288}{278} = 2\ 90705 \quad \text{and}$$

$$p = 807\ 33$$

The difference 807 82 — 807 33 = 0 49 mm Hg = 6 7 mm of water

Which of these formulas should be used for the calculations?

Numerous temperature measurements in shafts have shown that although for the first tens of metres or so the gradient Δt_{100} reaches 3 to 5°C and consequently approaches the figure for constant volume, at greater depths the value of the gradient varies between 0 and 1°C, with $\Delta t = aH$ approximately, the change of state of the air is polytropic Thus, careful observation of the change of temperature and pressure in two shafts at No 11 Smolianka mine, 707 and 711 m deep, made in 1937 by N K Tsolveg, showed that for the very wet downcast shaft, the polytropic index n averaged 0.9 in summer and 1 3 in winter; and for the upcast shaft also wet, it was 1.25 in summer and 1 4 in winter In dry shafts the polytropic index would be somewhat higher

Thus, the equation which best fits the facts is Equation (6-7)

However, since Equation (6-5) is much simpler than Equation (6-7) and since the value of the natural draught is only slightly affected by the choice of the equation for calculating the increase of pressure with depth, it is advisable to use Equation (6-5) since it is simpler

CHAPTER 7

PRINCIPAL LAWS OF AIR MOVEMENT UNDERGROUND

This chapter deals with the main assumptions and laws of mine aerodynamics which are needed for understanding the physical processes taking place when the air flows through the roadways. A special interpretation of the equation of Bernoulli, well known in hydraulics, is given, with deductions, the types of movement of air, the structure and types of air stream are reviewed, and the main laws of similarity are stated.

7-1. THE EQUATION OF CONTINUITY

Mass flow, that is, the mass flow of air per second to the different sections of mine workings in the absence of leakage, is constant

$$m_1 = m_2 = \ldots = \text{constant} \tag{7-1}$$

But $m = \frac{G}{g}$, where G = weight of air, equal to $vS\gamma$, where v = air velocity, S = cross-sectional area of the airway, γ = density

After substituting we obtain the following equation

$$G_1 = G_2 = \ldots = \text{constant} \tag{7-2}$$

from which it follows that *the weight of air discharged is also constant*

$$v_1 S_1 \gamma_1 = v_2 S_2 \gamma_2 \tag{7-3}$$

Equation (7-3) is called the *equation of continuity*, bearing in mind that $vS = Q$, where Q equals the volume of air flowing, instead of Equation (7-3) we can write

$$Q_1 \gamma_1 = Q_2 \gamma_2 \tag{7-4}$$

If $\gamma_1 = \gamma_2$, then $Q_1 = Q_2$, that is *the volume of air flowing is constant in conditions of constant density*. In this instance

$$\frac{v_1}{v_2} = \frac{S_2}{S_1} \tag{7-5}$$

that is, *the velocity in various cross sections of the airway when γ is constant is inversely proportional to the area of the cross section*

From Equations (7-3) and (7-5) we obtain

$$v_2 = v_1 \frac{S_1 \gamma_1}{S_2 \gamma_2} \quad \text{and} \quad Q_2 = Q_1 \frac{\gamma_1}{\gamma_2} \qquad \text{(7-6) and (7-7)}$$

The correction $\frac{\gamma_1}{\gamma_2}$ for the difference in densities of air can reach 8-10%, it must be introduced into calculations when an air survey of the mine is made (see below).

7-2. THE MAIN LAW OF MOVEMENT OF MINE AIR

The movement of air along the mine airways is subject to the general laws of aerodynamics

Positive factors which constrain the air to move along the mine airways include·

1 The work of the fan creating a head or depression, $h_f = p_1 - p_2$; under the effect of this suction or pressure the air moves from a space at high pressure p_1 to a space at low pressure p_2

2 The difference in pressures $\gamma_1 H_1$ and $\gamma_2 H_2$ of columns of air in vertical or inclined shafts, the pressure of the heavier column of air, for example $\gamma_1 H_1$, overcomes the pressure of the lighter column $\gamma_2 H_2$, forcing the latter out of the mine; in this case the stimulus for the movement of the air is the so-called *natural draught* h_d equal to $\gamma_1 H_1 - \gamma_2 H_2$

3 The difference between the velocity heads h_v' and h_v'' calculated from the average velocity v_{av} at the entrance to and at the exit from the mine $h_v' - h_v'' = \Delta h_v$ (for example, the wind blowing on the mouth of an adit)

Created by any method, the pressure difference is spent in overcoming the resistance of the mine workings to the air passing through them, let us denote this pressure difference expended in the mine h_r, then in the general case we obtain

$$h_f + h_d + \Delta h_v = h_r \qquad (7\text{-}8)$$

or if we develop the equation

$$p_1 - p_2 + \gamma_1 H_1 - \gamma_2 H_2 + k_1 \frac{v_1^2}{2g} \gamma_1 - k_2 \frac{v_2^2}{2g} \gamma_2 = h_r \qquad (7\text{-}9)^*$$

where the subscripts 1 and 2 refer to the points of entry and exit of air into and out of the mine, and k_1 and k_2 are the so-called coefficients of kinetic energy (see page 234)

* It is proposed that the pressure difference $p_1 - p_2$ should be converted from mm mercury to mm water

Equation (7-8) is known as *Bernoulli's equation*, it is the main equation for solving problems on the movement of gases and liquids and is fundamentally important for all hydrodynamics and aerodynamics The dimension of each of the members on the left-hand side of Equation (7-9) is kg/m^2, multiplying and dividing by metres, we obtain kg-m/m^3, that is, each member of Bernoulli's Equation (7-9) represents the *work done by a cubic metre of air while it moves through the mine workings* Dividing all the members also by γ, the density, and noting that m$^3\gamma$ is equal to weight, we discover that each member of Bernoulli's equation *expresses energy per unit weight, that is specific energy*

From Equation (7-9) it can be seen that the total energy contained in the air passing through the mine consists of three parts

(1) the *pressure energy* corresponding to the pressure drop or positive pressure created by the fan;

(2) the *potential energy* corresponding to the pressure difference created by the columns of air of height H of different specific gravity,

(3) the *kinetic energy* corresponding to the difference in velocity pressures Δh_v.

Bernoulli's equation obtained by us for the particular case of the movement of air in the mine is deduced in the mechanics of fluids for the movement of an *incompressible fluid* (gas) between two arbitrary cross sections *1* and *2* with subscripts 1 and 2 in Equation (7-9) referring to these sections. But air is compressible, it is well known that as air moves through the mine its specific weight under the effect of various factors, mainly temperature and pressure, continuously changes and consequently v, the specific volume, also changes Is it possible that Bernoulli's equation can be valid for this particular case?

It is explained in fluid mechanics courses that to write Bernoulli's equation in the form of Equation (7-9) with two different values of density γ in the initial and end sections is permissible in certain conditions. During the movement of the gas its compressibility can be neglected so long as its velocity is low in comparison with the velocity of sound (about 330 m/sec); for air, for example, the relative change of density at velocities of 100 m/sec is only 0 05 This velocity can be regarded as the limiting velocity at which it is permissible, with moving air, to use the laws of an incompressible fluid; the velocity of air flow through mine roadways, however, is considerably lower than this value.

In the particular case of the movement of air through deep shafts, the effect of compressibility of air becomes so large that it must be taken into consideration (see page 451).

Considering that the difference $\gamma_1 H_1 - \gamma_2 H_2$ and $h'_v - h''_v$ can be positive or negative, Equation (7-9) must be finally rewritten thus.

$$h_f \pm h_d \pm \Delta h_i = h_r \qquad (7\text{-}10)$$

Some Particular Cases of Bernoulli's Equation:

(1) With p_1 larger than p_2, when $p_1 = p_0$ (barometric), $v_1 = v_2$ and $\gamma_1 = \gamma_2$, $h_r = p_1 - p_2 = p_0 - p_2 = h_f$—*the case of exhaust ventilation by a fan*

(2) With $p_1 > p_2$, when $p_2 = p_0$; $\gamma_1 = \gamma_2$ and $v_1 = v_2$; $h_r = p_1 - p_2 = p_1 - p_0 = h_f$ — *the case of forced ventilation by a fan*

(3) With $p_1 = p_2$, and $\gamma_1 \neq \gamma_2$ and $v_1 = v_2$, $h_r = \gamma_1 H_1 - \gamma_2 H_2 = h_d$ — *the case of natural ventilation alone.*

(4) With $p_1 = p_2$, $\gamma_1 = \gamma_2$ and $v_1 \neq v_2$; $h_r = h'_v - h''_v = \Delta h_i$ — *the case of ventilation by the velocity pressure of the wind.*

(5) With p_1 greater than p_2, $\gamma_1 \neq \gamma_2$ and $v_1 = v_2$, $h_r = p_1 - p_2 + \gamma_1 H_1 - \gamma_2 H_2 = h_f + h_d$—*the case of ventilation by a fan with the natural ventilating pressure helping the fan.*

The kinetic energy coefficients k_1 and k_2 [see Equation (7-9)] in front of the velocity heads h'_v and h''_v take account of the inequality in distribution of the air velocity in the initial and final cross sections of the air flow, if the air velocity at all points of the cross section is uniform, then h_v should equal $\frac{v_{av}^2}{2g}\gamma$, in reality the velocity in the centre of the airway, because of the retarding effect of rough walls, is always higher than at the walls

Let the velocity at various points be equal to $v_{av} + \Delta v$, in which v_{av} is the average velocity equal to $\frac{Q}{S}$, and Δv is the deviation from this average velocity, it is known from arithmetic, that the average of the sum of the squares of $v_{av} + \Delta v$ is always greater than the square of the arithmetic mean, which is taken account of by the kinetic energy coefficient k

For rough pipes, $k = 1 + 213\alpha$, for main airways supported with three-piece timber frames, $k = 0.810 + 252\alpha$, in which α is the friction factor (see page 248)

For low air velocities and small differences between the initial and final cross sections of the roadway the third member of Bernoulli's Equation (7-8) is neglected since it is small.

Air always moves towards the lowest pressure, consequently the pressure of the moving air should continually decrease· with a forcing fan, the pressure created by it gradually falls, with an exhaust fan, the pressure drop increases as the fan is approached. This change in pressure can be measured by a water gauge connected to ducting at various points

7-3. DEDUCTIONS FROM BERNOULLI'S EQUATION

1. Let us assume that the air before entering the shaft is motionless, that is $v_1 = 0$, therefore Equation (7-10) for the total air flow in the mine will take the form

$$h_f \pm h_d = h_r + \frac{k_2 v_2^2}{2g}\gamma_2 = h_r + h_v'' \qquad (7\text{-}11)$$

where v_2 is the velocity of the air returning to the atmosphere

This equation shows that the pressure difference, created by the fan or by the natural ventilating pressure, if there is one, is expended in the following way

(a) in overcoming all the resistances to the movement of the air;

(b) in the creation of the velocity head h_v'' at the outlet to the atmosphere.

At a high velocity v_2 (which reaches 30-35 m/sec at the exit from the fan casing) the useless velocity head h_v'' can be larger than the useful head h_r

2. Let us assume, for the sake of simplicity, that there is no natural draught; therefore Equation (7-9) for the air flow in any mine working will be

$$p_1 - p_2 + k_1\frac{v_1^2}{2g}\gamma_1 - k_2\frac{v_2^2}{2g}\gamma_2 = p_1 - p_2 \pm \Delta h_v = h_r \qquad (7\text{-}12)$$

Since p_1 and p_2 are equal to the air pressure at the beginning and end of the airway under consideration, their pressure difference is the actual pressure drop measured by water gauge, or other instrument, Δh_v is the difference between the velocity heads at the beginning and the end of the airway; considering the values of v_1 and v_2, this difference may be equal to zero, or positive or negative, if $\Delta h_v = 0$, then $p_1 - p_2 = h_r$, that is the measured pressure drop of the section will be equal to the pressure lost in overcoming the resistance; if Δh_v is greater than 0, then evidently v_1 is greater than v_2, and S_1 is smaller than S_2, that is, the roadway widens in this instance

$$p_2 = p_1 - h_r + \Delta h_v \qquad (7\text{-}13)$$

instead of $p_2 = p_1 - h_r$ when $v_1 = v_2$ A comparison of these two equations shows that when the air velocity diminishes, v_2 being less than v_1, the air pressure p_2 at the point of reduced velocity is higher by the amount Δh_v The physical meaning of this correction becomes clear from the following. Each member of Bernoulli's equation, as explained above expresses the energy in unit weight of air, consequently, a reduction in the member $\frac{k_2 v_2^2}{2g}\gamma$ at the end of the section of air flow indicates a drop in kinetic energy, part of which in accord-

ance with the law of conservation of energy has been converted to pressure energy, numerically equal to p_2.

In a contraction of the air stream, that is with an increase of the air velocity, we obtain

$$p_2 = p_1 - h_r - \Delta h_v \qquad (7\text{-}14)$$

in this instance the reduction of p_2 is explained by the fact that part of the pressure energy has gone over to velocity energy This important law of aerodynamics can be formulated in the following way: **every increase in air velocity results in a drop in the air pressure, every drop in velocity results in an increase of air pressure.**

3 If the water gauge is connected to the drift of an exhaust fan and the air before it enters the mine is motionless ($v_1 = 0$), then the pressure p_2 in the fan drift (the case of an increase in velocity from 0 to v_2) will be expressed by Equation (7-13), and the pressure difference $p_1 - p_2$ measured by the water gauge or the pressure drop h_{exh} will be equal to

$$h_{exh} = p_1 - p_2 = h_r + h_v'' \qquad (7\text{-}15)$$

The term h_v'' is numerically equal to the velocity pressure $k_2 \frac{v_2^2}{2g} \gamma$ at the point of measurement of the pressure drop, calculated from the average air velocity at this point The physical meaning of this term, however, is completely different, in this instance it is not a head, but a pressure drop, resulting from the fact that the air from its motionless state ($v_1 = 0$) began to move with a velocity v_2, the total energy of the air flow is equal to the pressure energy plus the kinetic energy. Therefore in this instance it would be more correct to call the term h_v'' *the velocity head*

This conclusion can be formulated as follows the head h_{exh} measured at the fan drift is equal to the sum of the head h_r spent in overcoming the resistance of the mine workings and the velocity head h_v''

The head h_r is the mine head h_m From Equation (7-11) we obtain

$$h_m = h_{exh} - h_v'' \qquad (7\text{-}16)$$

that is, *the mine head (together with the fan drift) is equal to the head h_{exh} measured in the fan drift of an exhaust fan, less the value of the velocity head at the point of connection of the water gauge to the drift* At a high air velocity in the fan drift and a low mine head, the error from neglecting in calculations the correction for the velocity head will give a completely wrong idea of the value of the mine head.

During the measurement of the head in the fan drift of a forcing fan, there may be two cases

(a) the air velocities in the fan drift at the outlet from the fan and at the inlet from the atmosphere are equal, in this case the measured head h_f is equal to h_m;

(b) the velocity v_1 at the outlet from the fan is not equal to the velocity v_2 at the inlet and we propose to deduce Equations (7-17) and (7-18) according to Bernoulli's equation.

$$h_f = h_m - \left(k_1 \frac{v_1^2}{2g} \gamma_1 - k_2 \frac{v_2^2}{2g} \gamma_2\right)$$

or

$$h_m = h_f + \left(k_1 \frac{v_1^2}{2g} \gamma_1 - k_2 \frac{v_2^2}{2g} \gamma_2\right) \tag{7-17}$$

If the fan is fitted with an evase and the average velocity v_2 at the

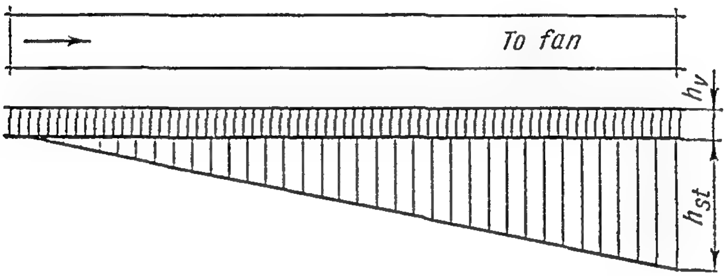

Fig 7-1 Graph of pressure drop

outlet from it is less than the velocity v_1 at the outlet from the fan casing, then a water gauge connected to the base of the evase will measure

$$h_f = h_{ev} - \left(k_1 \frac{v_1^2}{2g} \gamma_1 - k_2 \frac{v_2^2}{2g} \gamma_2\right) = h_{ev} - \Delta h_v \tag{7-18}$$

where h_{ev} = pressure lost in the evase; since v_1 is always larger than v_2, then Δh_v is positive; because h_{ev} usually is not large, the difference $h_{ev} - \Delta h_v$ can be negative, consequently at the beginning of the evase in this instance, there will be a pressure drop.

Pressure Graphs. A pressure graph is a curve showing the pressure changes in the course of the air flow Thus with an exhaust fan without leakages, the pressure along ventilation ducting will vary as shown in Fig. 7-1, in which the close shading indicates the velocity depression and the spaced shading indicates the pressure change caused by the overcoming of frictional resistance

7-4. TYPES OF AIR MOVEMENT UNDERGROUND

If we observe the slow movement of smoke-filled air it is easy to see that this flow consists of separate streams which do not intermingle; this type of movement is called *laminar* or *streamline flow*.

When the air velocity increases, the various layers begin to intermingle, their tracks become irregular, and no longer separately visible, this type of movement is called *turbulent* or *eddy flow*

Laminar flow is comparatively rare underground, e g in the movement of air through a layer of sand and stone dust in an airtight stopping Turbulent motion is much more frequent and usually all air flow in a working mine is turbulent Together with these two types of movement there is also an intermediate type (the region in which the breakdown of the laminar flow occurs is referred to as the transition), when air is seeping through bulkheads or stoppings made of planks or stone, or concrete, etc , also through ventilating doors, or the waste, or through packing which has not been built airtight, and through leaks in ventilation ducting

With increasing velocity, laminar flow passes through the intermediate state to turbulent flow The British scientist Reynolds showed in 1883 that this transition takes place when the Reynolds number (named in his honour) $Re = \frac{vd}{\nu}$ reaches the critical value of 2300. In this equation v = average velocity of the air, m/sec, d = any characteristic dimension for the cross section of this air flow, e g. for ducting, the diameter in metres, ν = kinematic viscosity, m/sec^2

When Re is greater than 2300, the flow is turbulent, when Re is less than 2300 it is laminar If appropriate precautions are taken, it is possible to extend the transition from laminar to turbulent flow up to very high values of Reynolds number (50,000 or more)

7-5. VELOCITY AND PRESSURE PULSATIONS IN THE AIR FLOW UNDERGROUND

By observing the air velocity at any point with some accurate instrument, such as a hot-wire anemometer and an oscillograph, it is easy to see that the velocity pulsates, changing with time In mine aerodynamics, this *micropulsation* is not taken into consideration, instead of the instantaneous values of the velocity, their *averaged* value is used.

Apart from these micropulsations, other larger pulsations are observed underground which are noticed, e g when a door is opened or closed, or when a train of mine cars passes or a cage or skip is hoisted, etc The change of velocity resulting from the pulsations can reach 40 to 50 per cent of the average velocity Special observations made by a mining engineer M. A Patrushev (Leningrad Mining Institute) at some mines in the Donets Basin have shown that:

(a) the value of the average air velocity fluctuates around a more or less constant average value;

(b) the change in the velocity of air flow is propagated almost instantaneously along its course (the observations took place in airways up to 1000 m long); this confirms once more the correctness of the assumption made above about the incompressibility of air (see pages 233 and 234) at the velocities usual in mines.

Figure 7-2*a* shows the micropulsations of the air flow velocity and the average value of the instantaneous velocities; Figure 7-2*b* shows

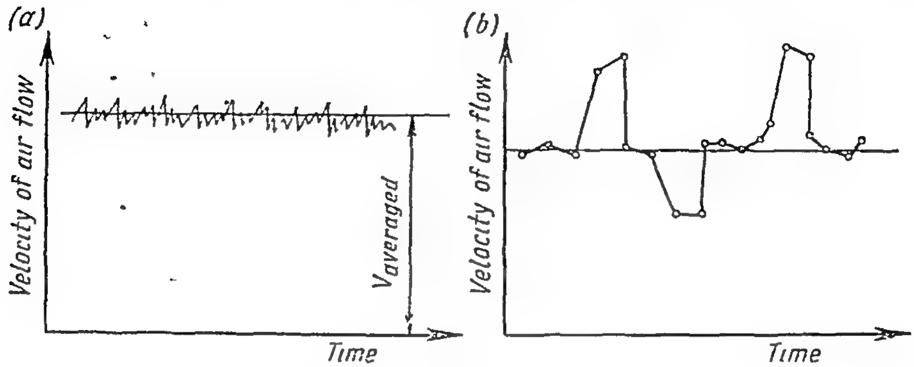

Fig 7-2 Air velocity pulsations

how the average velocity changes (and consequently also the air quantity) when a door is opened and closed, or a train of mine cars passes, etc

Apart from air velocity pulsations, there are also pressure pulsations underground; observations at several British mines have shown that these pulsations can be caused by

(a) the fan,

(b) the work of the hoisting plant,

(c) disturbance of the air flow pattern at points of sharp restriction of the roadways, and at the convergence of air streams at high velocity;

(d) the opening and closing of doors, and other incidents.

The fan creates pulsations of two types. small pulsations of 0 5 to 0.7 mm water with a frequency up to 4 c/s, and larger pulsations of 2 to 2 5 mm water, with a frequency of 1/3 to 1/4 c/s The value of the pulsations increases with increase of pressure loss, but with axial-flow fans, it also increases as the operating conditions of the fan approaches the hump of the characteristic

These pulsations are propagated against the direction of the air flow for a distance of up to 1000 m

The work of hoisting plant creates changes in the air pressure of the order of 1.8-2.0 mm water in large-diameter shafts and can produce considerably larger pulsations in small shafts; the frequency of

the pulsations agrees with that of the hoisting period and the pulsations occur at the foot of both shafts

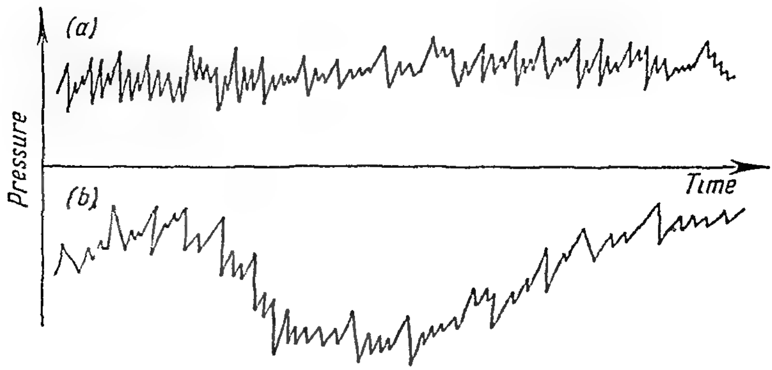

Fig 7-3 Pressure pulsations subject to the action of the fan

Figure 7-3 shows pressure pulsations from the fan (curve *a*) and as modified by the hoisting (curve *b*), the micro- and macropulsations are clearly seen

7-6. THE STRUCTURE OF AIR FLOW

If we take snap shots of the flow of fluids in which particles are floating, e g rosin, it is easy to notice next to the rough places and behind streamlined objects the eddies or rotating movements of the fluid These eddies occur also in air flow. Unlike solid bodies in which the rotational speed of the particles placed at the same radius remains constant, the eddying particles can rotate at various angular speeds

Turbulent flow is usually eddying According to modern views of turbulent flow there are two types of eddy (a) comparatively large eddies, and (b) small eddies

The large eddies are easily seen in water with the help of dyes; they receive a store of energy from an external source, e g the fan, at the moment of their formation, and carry this energy throughout all parts of the turbulent flow The smaller eddies, which fill the space between the large ones, dissipate the energy in the air flow, transforming it into heat.

7-7. TYPES OF AIR FLOW

In the movement of air underground, the following types of air flow exist

(1) *Air flow limited by solid walls* This is the commonest type of flow, occurring in shafts, roadways, and ventilation ducting.

(2) *Air flow without solid boundaries*, e.g. an air current entering a stope or leaving ventilation ducting (Fig. 7-4*a*); such currents are called *free streams*. If a free stream does not develop fully, i.e. at some distance from the original section is limited by the solid walls

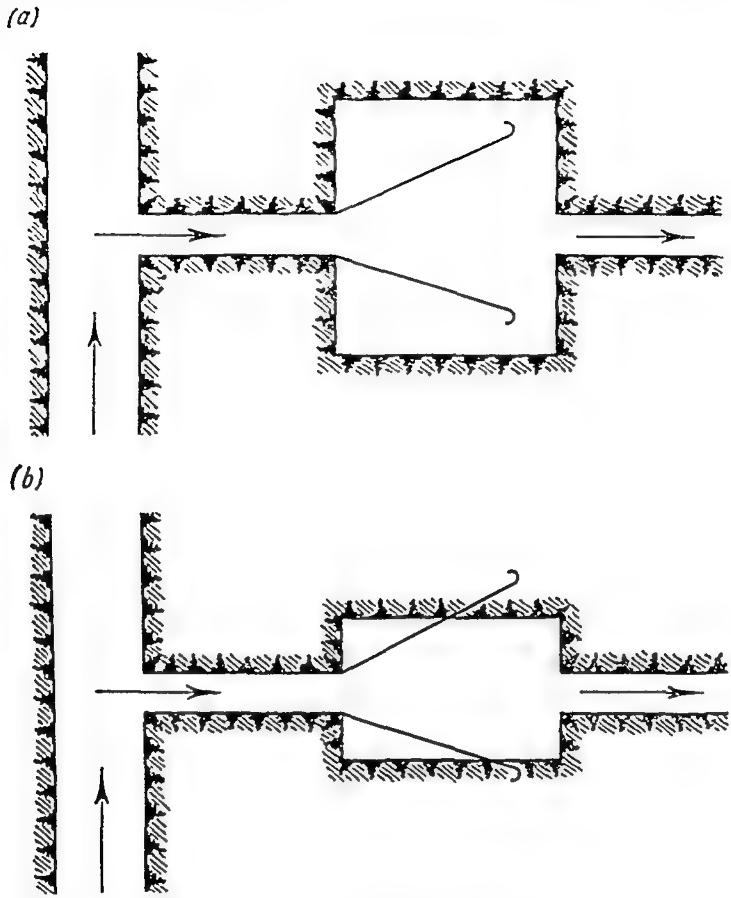

Fig 7-4 Free streams underground.
(a) complete; (b) incomplete

of the airway, it is called an *incomplete stream* (Fig. 7-4*b*); otherwise it is called a *complete stream*.

The application of the theory of free streams to the consideration of various problems of mine air flow has been extremely fruitful; it has enabled the complicated question of the stream lines around the timbering to be deeply studied. It has helped with the law of the variation of the concentrations of gases in the ventilation of dead ends after shotfiring, and provided formulas for the calculation of the air quantity needed in stopes after heavy blasting, etc. (see Sec. 15-7 and Part Three, Chap. 7).

7-8. THE LAW OF SIMILARITY

The value of the various coefficients of aerodynamic resistance in mines which will be discussed in the next chapter can be divided into full-size investigations in mines and laboratory studies in wind tunnels or involving models of mine airways. Model working is simpler, more convenient and less laborious

For the work on the model to be correct, the conditions of similarity must be observed In the most general case, these conditions include

(1) *Geometric similarity* The geometric similarity of the model and the prototype consists in a constant ratio between the geometrical values of length, area or volume in the model and in the prototype

$$\frac{L_m}{L_p} = \text{constant} \tag{7-19}$$

(2) *Kinematic similarity*, which implies that corresponding particles in the flow in the model and in the prototype must traverse corresponding distances in corresponding times

$$\frac{T_m}{T_p} = \text{constant} \tag{7-20}$$

For steady processes there is no need to insist on kinematic similarity.

(3) *Dynamic similarity*, i e the elementary forces directed towards the particles at corresponding points in the model and the prototype must be similarly directed and satisfy the equation:

$$\frac{P_m}{P_p} = \text{constant} \tag{7-21}$$

Strict observation of the conditions of similarity is not always possible, therefore in each instance only those factors are insisted on which are of decisive importance for the particular process Thus, for example, in hydraulics and aerodynamics it is considered that the main forces acting on the moving fluid are viscous friction (viscosity) and inertia, and therefore for similarity of the two flows it is necessary to ensure that the ratios of these forces in the model and the prototype are the same, i.e.

$$\frac{\text{force of inertia}}{\text{force of friction}} = \text{constant}$$

or

$$\frac{\rho \frac{v^2}{d}}{\mu \frac{v}{d^2}} = \frac{\rho v d}{\mu} = \frac{vd}{\nu} = \text{constant}$$

where v = air velocity
ρ = density of fluid (air)
d = characteristic linear dimension, e.g. diameter of ventilation ducting
μ = coefficient of viscosity
ν = kinematic viscosity.

The dimensionless number $\mathrm{Re} = \frac{vd}{\nu}$ in the last equation is called the Reynolds number as explained above.

Two flows are similar if they are geometrically similar and have the same Reynolds number, that is.

$$\mathrm{Re}_m = \mathrm{Re}_p = \frac{v_m d_m}{\nu_m} = \frac{v_p d_p}{\nu_p} = \text{constant} \tag{7-22}$$

With large Reynolds numbers, of the order of 80,000 to 100,000 or more, the forces of viscous friction in the moving fluid (air) become extremely small by comparison with the forces of inertia; the so-called *self-similar* condition of flow then occurs, in which mere geometric similarity is sufficient for the observance of similarity.

For model experiments in mine aerodynamics, air is the usual fluid.

In this case $\nu_m = \nu_p$ and Equation (7-22) takes the form:

$$v_m d_m = v_p d_p$$

or

$$\frac{v_m}{v_p} = \frac{d_p}{d_m} = M \tag{7-23}$$

That is, the velocity of a liquid or gas in the model is inversely proportional to the dimensions of the model; if the prototype is M times larger than the model, then the velocity of the fluid in the model must be M times faster than in the prototype.

The Reynolds number Re does not take into account the turbulence of flow, that is, the air velocity distribution across the ducting; experience has shown that the observance of geometrical similarity and equality of the Reynolds numbers are still not sufficient for obtaining completely similar results in experiments with two similar flows, *it is necessary also for the distribution of velocities in both flows to be similar*. Two further criteria must be observed to obtain this condition, these criteria were first formulated by V.N. Voronin: the first criterion is.

$$S'_K = \frac{l}{f} \tag{7-24}$$

the second criterion is:

$$S''_K = \frac{v}{v_{av}} \tag{7-25}$$

where l = characteristic linear dimension of the airway, e g. its radius or diameter

f = so-called scale of turbulence, equal to a_1R, in which a_1 = static constant (a_1 = 0.0032 to 0.0038) and R = = main scale dynamic measure of the flow which can be considered equal to the diameter

v = total averaged velocity at some point, e g in the centre of the airway

v_{av} = turbulent averaged velocity.

7-9. THE LAW OF DISTRIBUTION OF VELOCITY

As a consequence of the drag effect of the walls on the air flow, the air velocity gradually increases from the walls to the centre of the airway A curve showing the distribution of the air velocity in any cross section of an airway can be called the *velocity profile*; the ordinates of the curve represent the air velocity and the abscissas are distances from the wall of the airway.

The air velocity distribution in mine airways is, generally speaking, irregular and depends on the roughness of the walls. For airways with constant roughness, e.g timbered with frames, V N Voronin gives the following equation

$$\frac{v_{av}}{v_{max}} = \frac{1}{1+0\ 67\sqrt{\alpha/a_1}} = \frac{1}{1+11\ 8\sqrt{\alpha}} \tag{7-26}$$

where v_{av} and v_{max} = average and maximum air velocities

α = coefficient of friction

a_1 = static constant, equal to 0 0032

The rougher the walls are, the more the velocity profile will vary.

Next to local resistances, such as bends, restrictions, widenings, etc , and various obstacles to the air flow, such as mine cars, the velocity distribution becomes non-symmetrical in relation to the airway centre line

The air velocity at individual points is measured by Pitot tubes (see page 212) Recently, radioactive isotopes have begun to be used for measuring the velocity profile.

CHAPTER 8

AERODYNAMIC RESISTANCE OF MINE WORKINGS

8-1. THE LAW OF RESISTANCE

The resistances which the air overcomes in its movement through the mine are classified into the following categories:

(a) friction against the walls of mine workings;
(b) head resistance;
(c) local resistance

Before studying these three types of resistance, we must explain the law of resistance.

The *law of resistance* to the movement of air underground is the relation between the pressure drop Δp in the roads and the quantity Q or the flow rate v of the air passing through them.

Both theory and practice show that the main relation between v and Δp for laminar flow is the equation

$$\Delta p = Cv \tag{8-1}$$

and for turbulent flow,

$$\Delta p = Cv^x \tag{8-2}$$

in which x = 1.75 in completely smooth airways
x = 2 in rough airways such as those underground
C = a constant which is characteristic of the flowing fluid, the dimensions, and roughness of the airway.

A number of scientific investigations have authentically shown that turbulent air flow is subject to a square law ($x = 2$) or rather closely follows this law in all mine workings and ducting using an active ventilating current.

In the intermediate law, the exponent x is larger than unity but smaller than two.

The value of the exponent x for any fluid flow can be determined in the following way.

At the roadway section being investigated let the pressure drops be h_1 and h_2 corresponding to velocities v_1 and v_2 which are close to each other, then

$$h_1 = Cv_1^x \quad \text{and} \quad h_2 = Cv_2^x*$$

* It is assumed that with the change of flow rate from v_1 to v_2 the condition of flow does not change, i e $x_1 = x_2$.

Taking the logarithms of these equations, we obtain

$$\log h_1 = \log C + x \log v_1, \quad \text{and} \quad \log h_2 = \log C + x \log v_2$$

Subtracting the second equation from the first and solving for x, we obtain

$$x = \frac{\log h_1 - \log h_2}{\log v_1 - \log v_2} \tag{8-3}$$

The value of the exponent x can also be found in another way; plot the points A and B (Fig 8-1) with the ordinate axis log h and the

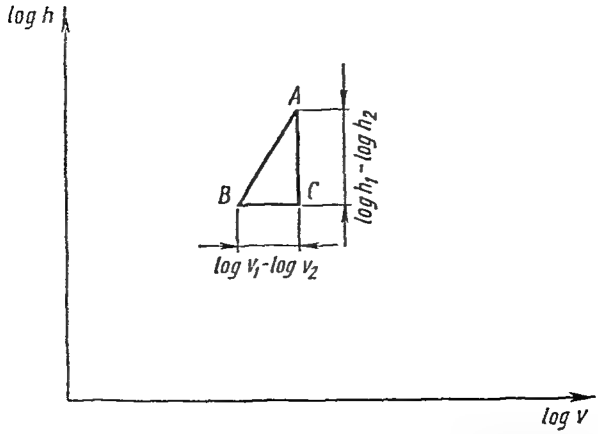

Fig 8-1 Diagram showing how to determine the exponent in the law of air flow underground

abscissa axis log v, having co-ordinates log h_1, log v_1, and log h_2, log v_2, the legs of the triangle ABC are evidently equal to $\log h_1 - \log h_2$ and $\log v_1 - \log v_2$; the tangent of the angle ABC i the required value of x.

The knowledge of the law of resistance alone, however, is not enough for finding the pressure loss h when air overcomes a resistance. It is necessary for this to deduce formulas which in every specific instance enable one, given the properties of the air and the nature of the resistance, to determine the value of the pressure loss h.

(1) *Resistance due to friction* Courses on hydraulics develop the following formula linking the quantity of fluid flowing through a a pipeline with the pressure head loss, the dimensions, and the properties of the pipeline

$$h = \lambda \frac{Lv^2}{D\,2g} \tag{8-4}$$

in which λ = a dimensionless coefficient of resistance

L and D = length and diameter of the pipeline, respectively

v = air flow rate.

Multiplying and dividing the right-hand side of Equation (8-4) by $\frac{\pi D}{4}$ we obtain

$$h=\frac{\lambda}{4}\frac{L\pi D}{\frac{\pi D^2}{4}}\cdot\frac{v^2}{2g}$$

or, knowing that $\pi D=P$ (perimeter) and $\frac{\pi D^2}{4}=S$ (cross-sectional area of the pipeline)

$$h=\frac{\lambda}{4}\frac{LP}{S}\frac{v^2}{2g}=\frac{\lambda}{8g}\frac{LP}{S}v^2=\alpha\frac{LP}{S}v^2 \tag{8-5}$$

in which $\alpha=\frac{\lambda}{8g}$.

A similar formula can be obtained for the movement of air through a mine, using the method of dimensions. The pressure loss due to the air flow through the pipeline depends on the following values: a dimensionless coefficient c describing the surface roughness of the pipeline and the physical parameters of the air (its viscosity μ and density ρ), the air flow rate v, the length, perimeter, and cross-sectional area of the airway

$$h=cf(\mu, \rho, v, L, P, S) \tag{8-6}$$

To reduce the number of values on the right-hand side of Equation (8-6), we shall relate the pressure loss to unit length L and unit perimeter P of the airway; since the pressure losses are evidently directly proportional to L and P (the product LP is the frictional surface of the airway), we can write

$$\frac{h}{LP}=cf(\rho, \mu, v, S) \tag{8-7}$$

Let the values ρ, μ, v, and S enter into Equation (8-7) raised to unknown powers α, β, γ, and δ, then

$$\frac{h}{LP}=c\rho^{\alpha}\mu^{\beta}v^{\gamma}S^{\delta} \tag{8-8}$$

Writing the dimensional equation for the left- and right-hand side of the Equation (8-8)

$$\left(\frac{\text{kg}}{\text{m}^2\ \text{m}^2}\right)=\left(\frac{\text{kg sec}^2}{\text{m}^4}\right)^{\alpha}\cdot\left(\frac{\text{kg sec}}{\text{m}^2}\right)^{\beta}\cdot\left(\frac{\text{m}}{\text{sec}}\right)^{\gamma}\cdot(\text{m}^2)^{\delta}$$

or

$$(R)^1\cdot(l)^{-4}(t)^0=\left(\frac{Rt^2}{l^4}\right)^{\alpha}\ \left(\frac{Rt}{l^2}\right)^{\beta}\cdot\left(\frac{l}{t}\right)^{\gamma}\cdot(l^2)^{\delta}$$

Reducing the right-hand side, we get

$$(R)^1(l)^{-4}(t)^0=(R)^{\alpha+\beta}(t)^{2\alpha+\beta-\gamma}(l)^{-4\alpha-2\beta+2\delta+\gamma}$$

Equating the powers, we obtain the following equations:

$$1=\alpha+\beta$$
$$-4=-4\alpha-2\beta+\gamma+2\delta$$
$$0=2\alpha+\beta-\gamma$$

Solving these equations, we obtain

$$\alpha=\gamma-1$$
$$\beta=2-\gamma$$
$$\delta=\frac{\gamma-4}{2}$$

Substituting in Equation (8-8) the values obtained for the exponents α, β, γ we have

$$\frac{h}{LP}=c\rho^{\gamma-1}\ \mu^{2-\gamma}v^{\gamma}S^{\frac{\gamma-4}{2}} \tag{8-9}$$

It has been found experimentally that with the usual values of Reynolds number Re in mines, the pressure losses are directly proportional to the square of the air flow rate, thus $\gamma = 2$; in this instance

$$\frac{h}{LP}=\rho^{2-1}\mu^{2-2}v^{2}S^{\frac{2-4}{2}}=c\rho\frac{v^2}{S}$$

and

$$h=c\rho\frac{LP}{S}v^2=2c\frac{\gamma}{2g}\frac{LP}{S}v^2=\beta\frac{\gamma}{2g}\frac{LP}{S}v^2 \tag{8-10}$$

hence

$$\beta=2c$$

In this formula, β is a dimensionless coefficient, which, as pointed out above, represents the roughness of the walls of the airway or mine working, it is called the friction coefficient or resistance coefficient, L is the length of the airway in metres, P and S are its perimeter and cross-sectional area, respectively, in metres and square metres, v = the air flow rate, m/sec; h = the pressure drop in mm H_2O or kg/m^2.

Usually the coefficient β and the expression $\frac{\gamma}{2g}$ in Equation (8-10) are combined into a single coefficient α, in view of the insignificant changes in the specific weight of the air while it moves through the mine

$$\alpha=\beta\frac{\gamma}{2g} \tag{8-11}$$

in which γ is the standard specific weight of air equal to 1.2 kg/m^3.

After substituting α for $\beta \frac{\gamma}{2g}$ in Equation (8-10), it can be rewritten in the following way:

$$h=\alpha \frac{LP}{S} v^2 \tag{8-12}$$

Because $v=Q/S$ in which Q=the quantity of air passing through the airway

$$h=\alpha \frac{LP}{S}\left(\frac{Q}{S}\right)^2=\alpha \frac{LP}{S^3} Q^2 \text{ kg/m}^2 \text{ or mm } H_2O \tag{8-13}$$

With $\gamma \neq 1.2$ kg/m³,

$$h=\alpha \frac{LP}{S^3} Q^2 \cdot \frac{\gamma}{1.2}$$

For circular airways or ducting

$$h=6.48\alpha \frac{L}{D^5} Q^2 \text{ mm } H_2O \tag{8-14}$$

The expression $\alpha \frac{LP}{S^3}$ in Equation (8-13) is called frictional resistance and is denoted by the letter R.

$$R=\alpha \frac{LP}{S^3} \tag{8-15}$$

Sometimes the resistance per unit length of airway, i e. the value $\alpha \frac{P}{S^3}$, is called specific resistance. From Equations (8-15) and (8-13) we obtain

$$h=RQ^2 \tag{8-16}$$

Equation (8-13) is extremely important for mine ventilation. It enables one to find the value of the head h which the fan must develop for the particular ventilating conditions of the mine. knowing the dimensions of the mine working, that is, their length L, cross-sectional area S, perimeter P, and the quantity of air Q, flowing through them, as well as the coefficient of friction α for the roughness of the walls of the workings.

Since. when mine ventilation is planned, the values of L, S, P. and Q are always given, the correctness of the calculation of the head depends in the final analysis on the reliability of the coefficient of friction α. It is obvious therefore how important the correct choice of the coefficient may be.

The manner of change in the coefficient of friction λ during the movement of a fluid in a pipeline is shown in Fig. 8-2. As can be seen, with low Reynolds numbers up to Re = 2,000 (log Re = 3 37)

the coefficient of friction is independent of the roughness of the walls of the pipeline and is equal to $\frac{64}{Re}$ (the Poiseuille law). With increasing flow rate, and transition into the intermediate and turbulent flow, the coefficient of friction begins to increase, the more so and the earlier, the greater the roughness of the walls. In Fig. 8-2,

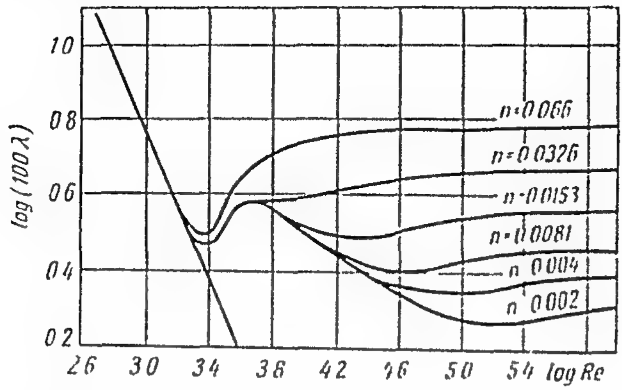

Fig 8-2 Curves of variation of coefficient of friction for fluids moving in pipelines at various values of Re

$n = \frac{k}{r}$ in which $k =$ the absolute roughness measured by the mean value of the projections from the walls of the pipeline; $r =$ the radius of the pipe.

In view of the law of similarity (Chapter 7) the value of the coefficient of friction and the way in which it changes, during air flow along similar airways, should be the same as in the movement of water Tests have confirmed this with the one difference that as a consequence of the higher roughness of mine supports, the transition from laminar to turbulent flow takes place earlier, at lower values of Re ($Re = 1{,}500$ and $\log Re = 3.17$) In addition, for the same reasons the values of the coefficients of friction are larger than shown in Fig 8-2 However, the nature of the curves is similar and is distinguished by the same "jump" in the region of the critical Reynolds number shown in Fig. 8-2

Of great practical importance is the problem of the constancy of the coefficient α with change of Re or, which is the same, with changing air flow rate in the mine workings

Numerous experiments carried out in the USSR to determine the values of coefficients of friction have confirmed that beginning with a Reynolds number $Re = 100{,}000$, the values of the coefficient of friction in mines become practically constant In roadways

of cross section of about 5 sq m, this value of Reynolds number corresponds to a flow rate

$$v=\frac{\mathrm{Re}\,v}{d}=\frac{100{,}000\ 14.4\ 10^{-6}}{2}=0.72\ \mathrm{m/sec}$$

Consequently in workings of 5 sq m cross section or larger (most common in modern mines), where the air usually moves at a rate of not less than 1 m/sec, the coefficient α can be considered constant; in secondary workings, where the air moves at lower flow rates, Re in individual cases may be less than 100,000, and therefore a somewhat reduced value of the coefficient α should be adopted. Considering the relatively small reduction in the coefficient α with the diminished value of Re, and the fact that the share of the secondary workings in the total resistance of the mine is generally small, we will assume for ventilation calculations that the value of the aerodynamic resistance of all mine workings is constant.

The small error resulting from this assumption makes the resistance and consequently also the head rather too large.

(2) *Head resistance* It was pointed out in Chapter 1 that the pressure of moving air on a stationary surface interposed in the flow is equal to $\frac{v^2}{2g}\gamma$ kg/m², in which v = the flow rate of the advancing air, m/sec The air pressure on a body of random shape placed across the air flow is proportional to the velocity head h_v, to the midsection S_{mid} (the midsection is taken to be the largest section of the body in a plane perpendicular to the flow) and to a coefficient c depending on the shape of the body, or

$$P_{head}=ch_vS_{mid}=\frac{cv^2}{2g}\gamma S_{mid}=c\rho\frac{v^2}{2}S_{mid}\ \mathrm{kg} \qquad \text{(a)}$$

The coefficient c is dimensionless, and is found experimentally. If the body is stationary it "resists" a force P directed head-on, therefore c is called the coefficient of head resistance

Some energy is naturally spent on passing the air flow around the body, the energy is mainly spent on the eddying which takes place downstream

Tests have shown that the coefficient of head resistance is subject to the same rule as the coefficient of friction, that is when the Reynolds number is high enough, it becomes constant and independent of the air flow rate.

If we divide the left-hand and the right-hand parts of Equation (a) by the cross-sectional area of the roadway, excluding the area of the body around which the air passes, that is by $S-S_{mid}$, we obtain

$$\frac{P_{head}}{S-S_{mid}}=c\frac{v^2}{2g}\gamma\cdot\frac{S_{mid}}{S-S_{mid}}\ \mathrm{kg/m^2}$$

The expression $\frac{P_{head}}{S-S_{mid}}$ is the pressure loss at one element of the supports The dimension of this value is kg/m², and denoting it by h, we can write

$$h_{head} = c\frac{v^2}{2g}\gamma\frac{S_{mid}}{S-S_{mid}} \text{ kg/m}^2 \text{ or mm } H_2O \tag{8-17}$$

but $R = h : Q^2$, consequently

$$R_{head} = \frac{h_{head}}{Q^2} = \frac{1}{Q^2}\cdot c\frac{v^2}{2g}\gamma\frac{S_{mid}}{S-S_{mid}}$$

but because $Q = v(S-S_{mid})$

$$R_{head} = \frac{c\ v^2\ S_{mid}\gamma}{2g\ v^2(S-S_{mid})^3} = 0\ 0612\frac{c\ S_{mid}}{(S-S_{mid})^3} \tag{8-18}$$

Thus if the value of the coefficient c is known, then the aerodynamic resistance of the body to the air flow can be calculated from Equation (8-18), it will increase as the cross-sectional area S of the roadway decreases and as the midsection of the body increases

In mine ventilation, head resistance to the air flow is offered by props, placed across the airway, and shaft equipment such as buntons and beams, as well as by props standing in the sides of the roadways If props or buntons are in line with each other. the value of their head resistance, when grouped in this way, will be the less, the closer they are together.

(3) *Local resistances* The conception of resistance to air flow can be extended also to such elements in the resistance of the mine workings as bends, restrictions, widenings, etc., which do not come into the category of frictional resistance, determined by the equation $R_{fr} = \alpha\frac{LP}{S^3}$, but nevertheless do involve a pressure loss when the air passes them They are called *local resistances.*

The pressure drop h, as the air passes a local resistance, is directly proportional to the velocity pressure h_v of the air and depends on the shape of the local resistance and not on its dimensions. In general, this loss is expressed by the equation

$$h_{local} = \xi\cdot h_v = \xi\frac{v^2}{2g}\gamma \text{ mm} \tag{8-19}$$

in which ξ = a dimensionless coefficient determined experimentally for each resistance

v = the air flow rate at the local resistance.

For some local resistances, design formulas are available, e g for sudden widenings.

It follows from Equation (8-19) that the coefficient ξ shows the share of the pressure expended on overcoming the local resistance in the velocity pressure.

The values of coefficients of local resistance underground generally depend only on the air flow rate.

The energy spent in passing the air through a local resistance goes into redistribution of velocity, and eddies in the wake of the local resistance.

We shall deduce the equation for calculating the resistance when the coefficient ξ of local resistance is known. To do this, we transform Equation (8-19).

Bearing in mind that $Sv = Q$, we obtain

$$h_{local} = \xi \frac{Q^2}{2g\ S^2} \gamma \tag{a}$$

By analogy with the resistance of a roadway, we can write

$$h_{local} = R_{local} Q^2 \tag{b}$$

and from (a) and (b), equating the right-hand parts of the equations and eliminating Q^2, we obtain

$$R_{local} = \xi \frac{\gamma}{2g} \frac{1}{S^2} = \xi \frac{1.2}{19.62} \frac{1}{S^2} = 0.0612 \frac{\xi}{S^2} \tag{8-20}$$

From Equation (8-20) we have

$$\xi = \frac{R_{local}}{0.0612} \cdot S^2 = 16.34 R_{local} S^2 \tag{8-21}$$

Equations (8-20) and (8-21) enable the value of the resistance R_{local} to be found from the area S, and the coefficient of local resistance ξ, and vice versa. These equations are particularly useful for the calculation of complicated resistances such as those in fan drifts, air crossings, etc., which consist of combinations of frictional elements with other resistances.

We must stress the difference between the coefficients ξ in Equation (8-19) and c in Equation (8-17), on the one hand, and the resistance R on the other; the values of the coefficients ξ and c are independent of the dimensions of the airway, for geometrically similar local and head resistances, the values of the coefficients ξ and c are constant. But the value of the resistance R depends on the dimensions of the airway, with increase of these dimensions, the value R diminishes in direct proportion to the square of the cross-sectional area, or to the fourth power of any linear dimension of the airway.

A comparison of Equation (8-20) for local resistances with Equation (8-18) for head resistances shows the similarity in their structure.

It is quite obvious that the total resistance of any roadway will be equal to the sum of its resistances, friction, head resistance and local resistances.

8-2. DETERMINATION OF THE COEFFICIENTS OF RESISTANCE

Coefficients of resistance can be determined directly underground or in the laboratory Mine investigations are more reliable but they are not always possible for various reasons; laboratory investigations have the advantage that they allow the conditions of the experiment (the air flow rate, type of support, etc.) to be changed in wide limits

The method of determining coefficients of resistance is simple The following are determined (1) the pressure drop when the air passes through the required resistance, and (2) the quantity of air passing through this resistance Then, depending on which resistance is to be determined, one of the Equations (8-13), (8-17), or (8-19), is solved for the required coefficient of resistance, and the measured pressure drop and air quantity are substituted in it, as well as the values which control the particular resistance (L, P, and S). To measure the pressure drop between the beginning and the end of the experimental length, a micromanometer is used with Pitot tubes installed at these sections and connected by rubber tubes to the micromanometer The air quantity is determined by the method of small areas (p 215) or, if underground by an anemometer, by measuring in front of the observer or "in the section". In the determination of the coefficients of local and head resistances, the pressure drop due to friction must first be subtracted from the measured pressure drop in the experimental length; the value of the coefficient of friction in the experimental length must therefore be determined first.

8-3. COEFFICIENTS OF FRICTION

In this method, the values of the coefficient of friction are obtained purely empirically Recently, however, in the USSR, various equations for calculating the coefficient of friction have been proposed Thus, for roadways with smooth concrete sides, V N Voronin has proposed the following expression for Reynolds numbers from 4,000 to 4,000,000

$$\lambda = \frac{0\ 28}{\log^2 (0\ 1\ \mathrm{Re})} \tag{8-22}$$

for $\lambda = 4$, $\beta = 65\ 4\alpha$

For timbered roadways, V N. Voronin, applying the theory of free streams to the air flow through the intervals between the frames, has deduced the following expression·

$$\alpha\ 10^4 = \frac{1}{\left(0\ 21 + 0\ 11 \log \frac{\varepsilon}{\Delta\ m_1\ m_2}\right)^2} \tag{8-23}$$

in which ε = the so-called cross-sectional gauge of the timber, equal to $\frac{0.48\sqrt{S}}{d}$

S = area of roadway, sq m

d = diameter of timbers, m

Δ = longitudinal gauge (spacing/diameter ratio)

l = spacing between centres of timber frames, m

$$m_1 = 1 + \frac{0.07}{\Delta} - \sqrt{\frac{0.14}{\Delta}}$$

m_2 = the ratio between the timbered part of the perimeter and the total perimeter.

An equation of similar structure was proposed by V V. Kashibadze for roadways with steel supports (Appendix 2).

Equation (8-23) is complicated and unsuitable for practical calculations. The much simpler equation of L.D. Voronina can be recommended:

$$\alpha = \sqrt{a + b\frac{l}{S}} \tag{8-24}$$

in which l and S have the same significance; a and b are numerical coefficients; the values of these coefficients for mines timbered with three-piece sets are: $a = 75$, $b = 542$; for roadways timbered with four-piece sets: $a = 55$, $b = 863$.

The design equation for roadways lined with concrete, concrete blocks, or brickwork, and of circular cross section is:

$$\alpha = \frac{0.015}{\left(1.74 + 2\log\frac{D}{d_0}\right)^2} \tag{8-25}$$

in which D = the diameter of the roadway, m

d_0 = the height of the projections from the rough surface of the supports, m (see Table 8-1).

For roadways which are not of circular cross section

$$\alpha = \frac{0.015}{\left(1.74 + 16.3\log\frac{4.8\sqrt{S}}{d_0}\right)^2} \tag{8-26}$$

in which S = the cross section of the roadway, sq m.

Values of the coefficient of friction for mine workings with various types of support are given in Appendix 2 A number of scientists who investigated the effect of various factors on the value of the coefficient of friction established the following points: (a) the coefficient of friction increases with increase of the diameter

TABLE 8-1 Values of Surface Roughness, d_0, of Airway Linings

Type of lining	Linear dimension of projections from surface, d_0, metres
Smooth concrete	0.00025
Rough concrete	0 00070
Brickwork	0 00130
Rubble masonry	0 00800
Rough rubble masonry	0 02000

of the timber if the cross section of the working is constant; (b) the coefficient of friction diminishes with increasing cross-sectional area of the working if the timbering remains of the same size; (c) in workings supported with timber sets, it increases with increase of spacing/diameter ratio up to the value $\frac{l}{d} = 5\text{-}6$, and then diminishes (Fig 8-3); the larger values of α apply to timbers of larger diameter; (d) change of air flow rate or of Reynolds number has the following effect on the coefficient of friction as pointed out above, α can be considered constant beginning with Reynolds number of about 100,000 ($\log Re = 5$), with smaller Re, the coefficient of friction at first increases slightly (Fig 8-4) and then in the transitional area (from turbulent to laminar flow) suffers the changes shown in Fig 8-4 according to tests by Professor P I. Mustel, these tests showed that if models of timber supports were tested in a wind tunnel, when the air flow reaching the timbered length was an undisturbed flow coming from an untimbered length, the change in the value of α followed the curve b

The curves obtained by Professor P I Mustel for the air passing through equipped shafts (Fig 8-5) are of great interest. The zig-zag shape of the curves in the transition area is explained by the fact that the air in the various compartments moves independently at different flow rates and consequently with gradual reduction of the average flow rate the different air flows do not pass through the transitional area simultaneously, with the result that the general curve relating α to the Reynolds number has humps and valleys

Change in the value of the coefficient of friction with change in the spacing/diameter ratio Δ is explained in the following way in accordance with the theory of free streams the air, leaving the restriction of the timber frame, expands at a certain angle (Fig. 8-6); then, depending on the value of the spacing/diameter ratio, the free stream, not fully developed, is again restricted (at values of Δ

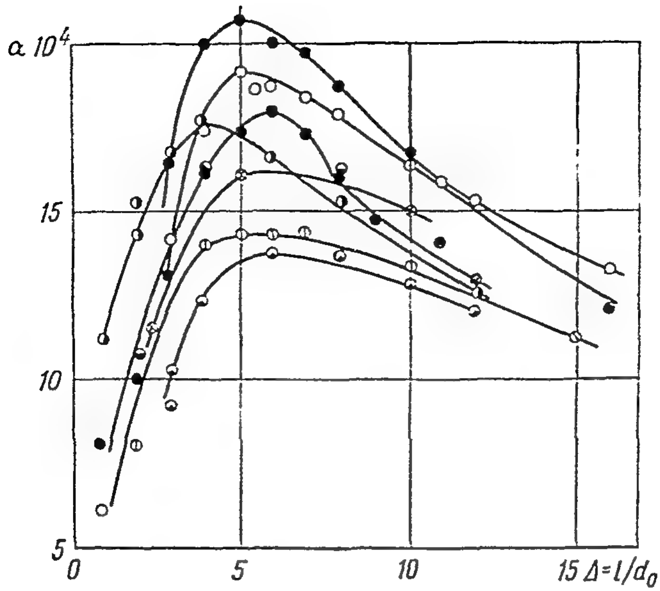

Fig 8-3. Curves showing relationship between α and l/d (spacing/diameter ratio of the timbering) for main airways using timbers of various diameter, d

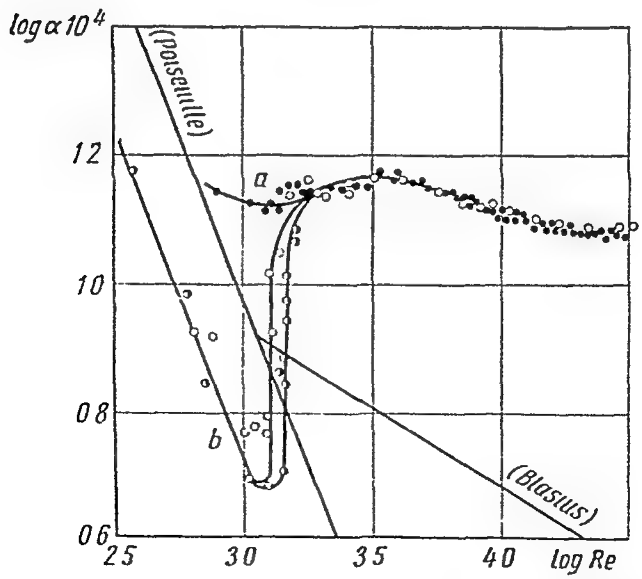

Fig 8-4. Graph of the change in the coefficient α of main airways in relation to Reynolds number

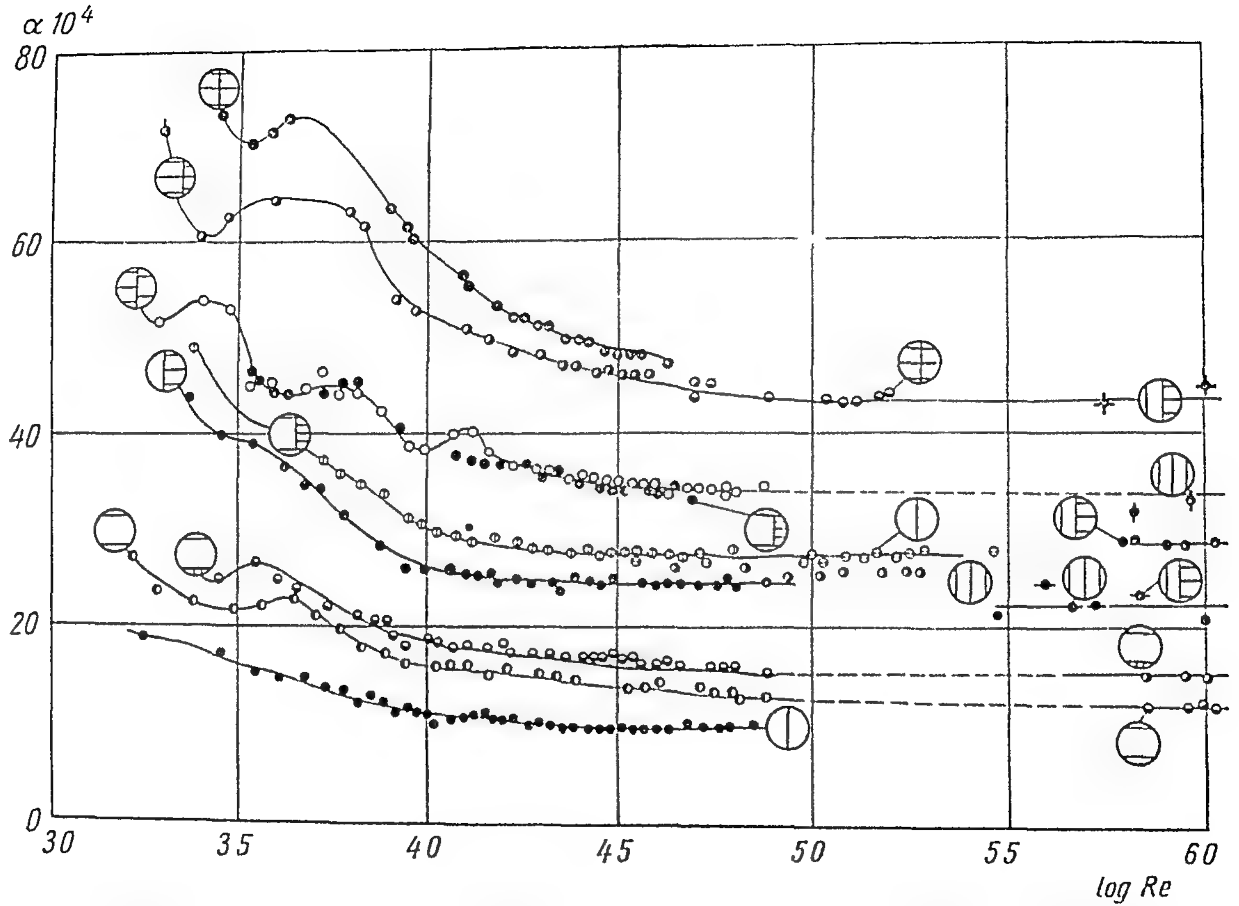

Fig 8-5 Graphs of variation in the coefficient α of circular shafts in relation to Reynolds number (according to P I Mustel)

from 4 to 6); or, expanding to the maximum value, it moves for some distance in contact with the roadwork walls (at values of Δ larger than 6); and because each enlargement of the stream is accompanied by loss of energy (pressure head) which increases with the amount of the enlargement, the end result is that the coefficient of friction depends basically on the degree of enlargement of the flow and on the number of expansions per unit length of the working.

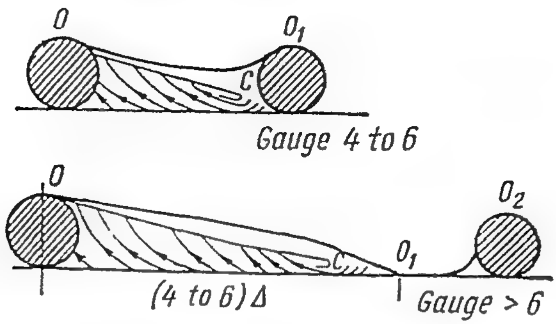

Fig. 8-6. Diagram of the air stream between neighbouring timber frames

With a small spacing/diameter ratio, the expansion effect is the greater, with a large spacing/diameter ratio, the spacing of the timber frames has the predominant effect.

Fig. 8-6, line *OC*, shows the boundary of the free stream.

The value of the coefficient of friction also depends on the friction of the moving air against the still air in the spaces between frames, and against the walls, as well as on the structure of the flow.

To reduce the aerodynamic resistance of mine workings, they should, so far as possible, be built in a straight line or with smoothly rounded curves. The supports should be correctly built so that individual sets do not project into the working, no unlagged spaces should be left behind the timbering, and all sets should be stayed and tightened. To reduce their resistance, timber sets are covered with planks or slabs of timber, either continuously, or staggered (Appendix 2, p. 562). With metal or precast concrete supports, the intervals between the frames are packed with timber or special concrete slabs, or the frames are built with the wide surface facing the working.

8-4. COEFFICIENTS OF LOCAL RESISTANCE

The local resistances which are specific to mine ventilation include air crossings, doors with regulators in them, moving or standing mine cars and trains, bends in the fan drift, and objects which encumber the workings.

As pointed out above, the air pressure lost at a local resistance can be calculated by the formula

$$h_v = \frac{v^2}{2g} \cdot \gamma \cdot \xi$$

The value of the coefficient of local resistance depends on the roughness of the walls of mine workings. Thus, for right-angled bends, V. B. Komarov found the following empirical formula

$$\xi_{rough} = \xi_{smooth} + 235\,(\alpha_{rough} - \alpha_{smooth}) \tag{8-27}$$

in which ξ_{rough} and ξ_{smooth} are the coefficients for rough and smooth airways, respectively, α_{rough} and α_{smooth} are the corresponding coefficients of friction.

A.A. Kharev gives a similar formula for a right-angled bend

$$\xi_{rough} = (\xi_{smooth} + 280\alpha_{rough}) \cdot \frac{b}{H} \tag{8-28}$$

in which H = the height of the airway
b = width of the airway.

The values of coefficients of local resistance are given in Appendix 3.

8-4.1 Methods of Reducing Coefficients of Local Resistance

When the air flow rate is high at the local resistance, the pressure loss can be considerable, and the power lost in overcoming the local resistance will also be considerable. Indeed, at a flow rate of 5 m/sec, h_v = 1.5 mm H_2O, at a flow rate of 10 m/sec, h_v = 6 mm H_2O, and at a flow rate of 15 m/sec, h_v = 13 5 mm H_2O. Let us assume that 20 m^3/sec of air pass through this local resistance, then assuming the coefficient of local resistance to be 2, and the efficiency of the fan 0.6, the annual power consumption to overcome this local resistance at 10 m/sec will be $\frac{6 \times 2 \times 20 \times 24 \times 365}{102 \times 0.6} = 34{,}400$ kWh, and at 15 m/sec 78,300 kWh. Naturally, this calls for reducing the coefficient ξ.

The coefficient ξ can be reduced·

1. At junctions of workings by (a) rounding sharp corners; (b) rounding off junctions; (c) chamfering internal sharp edges; (d) placing guide vanes (see Appendix 3); (e) building smooth linings at junctions;
2 At air crossings by building a smooth intake, and especially a smooth outlet from the crossing;
3. At points of sudden enlargements or restrictions by building smooth intakes or diffusers

8-4.2 Calculation of Local Resistances in Ventilation Design

As pointed out above, local resistances represent a large extra power cost in ventilation only when the value of the coefficient of local resistance is high, the air flow rate is high, and the air quantity passing the local resistance is also high; such locations include the fan drift (considering it as a point of complicated local resistance), air crossings, or bends in mine workings which take large air flows. All the other local resistances, although numerous, are generally disregarded.

A simplified method has been accepted for the calculation of the overall mine depression (see later, Part Three) because in the formula

$$h = \alpha \frac{LP}{S^3} Q^2$$

for calculating the depressions of the separate workings the symbol Q is generally replaced by the planned quantity of air; actually, leakages underground reduce the air quantity to below this level and consequently the true depression is less than that calculated. Thus, by neglecting the local resistances we err by understating the depression; by neglecting the leakages we err by overstating it. Approximate calculations made by the authors have shown that these errors more or less cancel each other out.

The resistance of sudden widenings and restrictions are only calculated for the depressions of rock salt mines, potash mines, and others which are worked by stoping; because of the large dimensions of stopes, their frictional resistance is negligible, but the local resistances at the air entry and outlet can be considerable.

The possibility is not excluded, however, that with more accurate coefficients of friction, with increase in air flow rates underground, and with reduction in leakages, it may be necessary to calculate local resistances in working out the depression in collieries and metal mines, either by introducing a general correction factor to the total pressure loss or by approximately estimating the local resistances.

8-5 HEAD RESISTANCE

The greatest head resistances which are met with underground are found at the mine shaft buntons; as depths increase to 1,000 m or more, the depression in the shaft caused by the increased head resistance of equipment, such as buntons, begins to occupy an increasingly large share of the total mine depression Head resistance is also offered by props which stand across the roadway, and the supports which carry an axial-flow fan.

To reduce head resistances, the bodies which offer them are streamlined by rounding the forward part facing the air stream and placing a sharp point on the rear side. Fig. 8-7 shows some of these

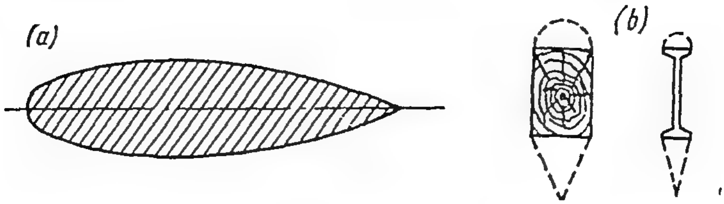

Fig 8-7 Shapes
(a) streamlined, (b) examples of streamlined shaft buntons

streamlined shapes Tests have shown that the aerodynamic resistance of streamlined equipment is 40 to 50 per cent less than that of ordinary equipment.

The values of coefficients of friction in shafts with streamlining are given in Appendix 2.

8-6. THE EFFECT OF VARIOUS FACTORS ON THE VALUE OF AERODYNAMIC RESISTANCE

8-6 1 The Effect of Change of Air Flow Rate

(a) Frictional resistance and local resistance at Reynolds numbers of the order of 150,000 to 200,000 or more, as has been mentioned, are practically constant, at smaller Re numbers they at first diminish and then, beginning with Re = 1,000 they abruptly increase (Fig. 8-5).

(b) Head resistances maintain a constant value in the interval between $\mathrm{Re} = 10^4$, and $\mathrm{Re} = 10^{5\ 5}$ corresponding to flow rates between 1 5 and 15-20 m/sec. With further increases of Reynolds number, the values of the coefficients of resistance rapidly fall, with diminution of Re they gradually increase.

8-6.2 Effect of Change in Specific Weight of Air

Since $\alpha = \beta \frac{\gamma}{2g}$, as explained above, it is natural that with change in specific weight of air, the aerodynamic resistance also changes; the resistance of airways through which cold air passes at sea level will be larger than the resistance of the same airways passing hot air at an altitude of 1,500 to 2,000 m

8-6.3 Effect of Water Dripping Down a Mine Shaft

The largest amount of water is usually observed dripping down the upcast shaft, where in addition to leakages through the shaft lining, there are leakages out of the drainage column, and moisture which has condensed out of the mine air from the upper part of the shaft

Depending on the upward velocity of the air in the shaft and the downward velocity of the droplets of water the following may happen (a) the largest drops of water fall down the shaft, overcoming the pressure of the upward current, and reach the bottom; (b) the medium-sized droplets of water remain suspended in the shaft; (c) the very smallest drops of water are taken up by the air current into the fan drift where part of them falls out with the air. The water flowing back into the shaft naturally increases its resistance; to reduce the harmful effect of this water the fan drift should be fitted with a cross gutter from which the water can be pumped away Also at the point where the shaft joins the fan drift a low ridge can be built; these simple measures have allowed the air flow through a number of deep South African mines to be greatly increased

The fact that much of the water is suspended in the shaft is proved by the following observation· after the fan had been stopped, for several minutes a short but violent downpour of rain took place. This water also increases the resistance of the shaft; to eliminate it, the fan speed (rpm) should either be slightly increased or slightly reduced; in both instances the output of the fan will increase. The critical air flow rate in the shaft is somewhat higher than 10 m/sec.

In very deep shafts, e g of 1,500 m, the specific weight of the air at temperatures of 27 to 28°C (if air conditioned) and a pressure of $760+\frac{1{,}500}{100}\times 9.5=902.5$ mm Hg will be equal to $\frac{0.455\times 902.5}{300}=$ $=1.365$ kg/m^3, i.e. almost 14 per cent above the normal specific weight of 1.2 kg/m^3, consequently the resistance of the workings will be 14 per cent higher.

A sharp drop in the specific weight of the air can be expected in a mine fire when, with an air temperature of about 1000°C, the specific weight of the air diminishes down to 0.3 kg/m^3 or even less, and in certain conditions this can spontaneously reverse the air current.

8-7. UNITS OF RESISTANCE

In Section 1 of this Chapter it was pointed out that the value of the resistance of roadway can be taken from the equation

$$R=\alpha\frac{LP}{S^3}$$

and the relation between the resistance, the pressure, and the air quantity from the equation $R = \frac{h}{Q^2}$. From this it can be seen that to obtain the value of the resistance of any roadway the pressure loss in the airway is divided by Q^2, the square of the air quantity passing through it This is a general statement and applies also to head and local resistances.

With $h = 1$ and $Q = 1$, $R = 1$, i.e a roadway with unit resistance has 1 m³/sec of air passing through it with a pressure of 1 mm H_2O, or in general $h/Q^2 = 1$

The dimensions of the unit of resistance are as follows·

$$(R) = \left(\frac{h}{Q^2}\right) = \frac{\text{kg/m}^2}{(\text{m}^3/\text{sec})^2} = \frac{\text{kg sec}^2}{\text{m}^8} \tag{8-29}$$

This unit is adopted in the Soviet Union under the name of *kilo-murg*, after the French investigator Murgue

The kilomurg is often too large a unit for measuring mine resistances, the resistance of a roadway 50 m long and 6 sq m cross section being equal to 0 0025 kilomurg Therefore, along with this basic unit of resistance, another unit, one thousandth of its value is used, which is called the *murg* and is denoted by the Greek letter μ Obviously 1 kμ = 1,000 μ The resistance of the roadway mentioned above will be 2.5 murgs = 2 5 μ.

Apart from murgs, other units of resistance are used in mines in other countries Thus, in Germany the *Weisbach* is used (w) which has the same value as the kilomurg, and in Britain the *Atkinson* is used, equivalent to 0 0061 kilomurg, a resistance of one Atkinson is offered by a roadway in which the pressure drop from 1,000 ft³/sec of dry air is equivalent to 1 inch of water gauge, at 15 5°C and 762 mm mercury.

The value of local and head resistances is also expressed in kilomurgs and murgs

It is necessary to emphasize that the value which indicates the difficulty or the ease with which a roadway can be ventilated is not the coefficient of friction but the resistance R Roadways can have the same coefficient of friction but if their cross sections are different, the larger roadway will be easier to ventilate than the smaller.

Table 8-2 shows, in murgs, the values of the resistances of 100-m lengths of roadways timbered with three-piece sets for various cross-sectional areas, with $\alpha = 0\ 0015$, for other values of the coefficient of friction the values of a correction factor k are given in the final column.

TABLE 8-2 **Resistances of 100-metre Lengths of Roadways, Murgs**

S, m²	Tenths of a square metre										α	k
	0	1	2	3	4	5	6	7	8	9	0.0010 0.0011	0.67 0.73
2	110	97.5	86.9	77.7	70.0	63.0	57.2	52.1	47.5	43.6	0.0012	0.80
3	40	36.9	34.1	31.5	29.3	27.2	25.4	23.7	22.2	20.8	0.0013	0.87
4	19.5	18.1	17.3	16.3	15.4	14.5	13.8	13.0	12.3	11.7	0.0014	0.93
5	11.2	10.7	9.95	9.59	9.22	8.82	8.42	8.07	7.72	7.42	0.0015	1.00
6	7.08	6.81	6.53	6.28	6.03	5.80	5.58	5.38	5.18	4.98	0.0016	1.07
7	4.82	4.65	4.49	4.20	4.06	3.92	3.80	3.80	3.67	3.55	0.0017	1.13
8	3.45	3.34	3.23	3.15	3.06	2.97	2.88	2.80	2.72	2.64	0.0018	1.20
9	2.57	2.57	2.43	2.36	2.30	2.24	2.19	2.14	2.08	2.03	0.0019	1.27
10	1.97	1.93	1.88	1.79	1.75	1.70	1.67	1.67	1.63	1.59	0.0020	1.33

Example. Determine the resistance of a roadway length of 6.6 sq m cross section, 600 m long, with $\alpha = 0.0017$

Solution. $R = 5.58 \times \frac{600}{100} \times 1.13 = 37.8$ murgs

8-8. EQUIVALENT ORIFICE

The equivalent orifice is a conception which gives a convenient visual presentation of the relative ease of ventilating a mine.

The equivalent orifice of a mine is an opening A in a thin plate, through which the same air quantity passes under the effect of a pressure difference between the sides of the plate, as would pass through the mine under the same head.

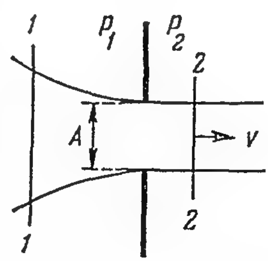

Fig. 8-8. Air flow through an opening in a thin plate

Let us determine the area of an equivalent orifice Let a certain air quantity, Q m³/sec, flow through the orifice in a thin plate of area A sq m (Fig. 8-8); let the pressure on the left-hand side of the plate be p_1 and on the right-hand side p_2 kg/m²; $p_1 - p_2 = h$, the mine head. For cross section *1-1*, far enough from the opening for the original flow rate v_1 to be neglected, and for cross section *2-2* at a point in the stream where the streamlines are parallel, we will use Bernoulli's equation for an incompressible gas.

$$p_1 + \frac{v_1^2}{2g}\gamma = p_2 \frac{v_2^2}{2g}\gamma$$

from which, because $v_1=0$,

$$p_1-p_2=h=\frac{v_2^2}{2g}\gamma$$

and consequently

$$v_2=\sqrt{\frac{2gh}{\gamma}}\ \text{m/sec and } Q=v_2A_2$$

in which A_2 = the cross section of the stream in the plane *2-2*. But $A_2 = \varphi\cdot A$ in which φ = the coefficient of discharge which on the average equals 0.65

Consequently

$$A=\frac{A_2}{\varphi}=\frac{Q}{\varphi\ v_2}=\frac{Q}{\varphi\sqrt{\frac{2gh}{\gamma}}}=\frac{1}{0.65\sqrt{\frac{19.62}{1.2}}}\ \frac{Q}{\sqrt{h}}=\frac{0.38Q}{\sqrt{h}} \qquad (8\text{-}30)$$

The dimensions of the equivalent orifice are

$$[A]=\frac{Q}{\sqrt{gh\gamma^{-1}}}=\frac{\text{m}^3/\text{sec}}{\sqrt{\text{m/sec}^2\ \text{kg/m}^2\cdot\text{m}^3/\text{kg}}}=\text{m}^2$$

The simple dimension of the equivalent orifice, square metres, enables the value of the orifice to be visualized very simply and thus to judge the degree of ease or difficulty of ventilating a colliery or metal ore mine. The larger the value of A, the "wider" is the mine, and the easier it is to ventilate. Metal ore mines have been conventionally classified into mines with an equivalent orifice less than 1 sq m, which are "narrow", those with A = 1 to 2 sq m, which are "medium", and those with A above 2 sq m, which are "wide".

To determine the value of the equivalent orifice (or resistance) of an operating mine, the following procedure is used. A rubber tube is lowered down the upcast shaft, with its upper end connected to a water gauge installed. (a) outside the mine building, or (b) in the mine building; in the latter case the other arm of the water gauge must be connected by a rubber tube to the space outside the building

The rubber tube must be lowered into the shaft to a point above the fan drift. The measured pressure drop will be equal to the mine head h_m

The quantity of air passing through the mine, Q_m, is measured: (a) in a vertical shaft, at the pit bottom, the air quantity being subsequently reduced to the standard specific weight of 1.2 kg/m^3; (b) in an inclined shaft, in the shaft itself not nearer than 10 to 15 m from the fan drift

$$A_m=\frac{0.38Q_m}{\sqrt{h_m}} \quad\text{and}\quad R_m=\frac{h_m}{Q_m^2}$$

In connection with Equation (7-30) we must point out that since the natural ventilating pressure nearly always operates in mines (Chapter 10) and helps or hinders the work of the fan, the equivalent orifice should be calculated from the equation

$$A=0\ 38\frac{Q}{\sqrt{h_m \pm h_n}}$$

and the mine resistance should be calculated according to the equation

$$R=\frac{h_m \pm h_n}{Q^2}$$

If the natural ventilating pressure is ignored, this implies increasing or reducing the values of A and R

Because $h=RQ^2$, Equation (8-30) can be rewritten

$$A=0.38\frac{Q}{\sqrt{RQ^2}}=\frac{0\ 38}{\sqrt{R}} \tag{8-31}$$

$$R=\frac{0\ 144}{A^2} \tag{8-32}$$

in which R = the mine resistance in kilomurgs; if R is expressed in murgs, then

$$A=\frac{0\ 38}{\sqrt{\frac{R}{1\ 000}}}=\frac{12}{\sqrt{R}}\ \mathrm{m}^2 \tag{8-33}$$

and

$$R=\frac{144}{A^2}\ \text{murgs} \tag{8-34}$$

If we consider Equations (8-31) to (8-34), we can conclude that

1 The value of the resistance and the equivalent orifice of a mine or roadway depends solely on the dimensions of the roadways and their coefficients of friction, and not on the quantity of air flowing; change in the cross-sectional area S of the roadway especially affects the values of R and A.

2 The resistance R is directly proportional to the length of the roadway, but the equivalent orifice is inversely proportional to the square root of this length; with increase of length of the roadway two-, three- or five-fold the resistance increases correspondingly; this relationship is simpler than that for the equivalent orifice· the equivalent orifice would be correspondingly reduced $\sqrt{2}$, $\sqrt{3}$, or $\sqrt{5}$ times.

3. The relationship between the pressure drop in the roadway and its cross-sectional area is also more simply expressed through the resistance than through the equivalent orifice.

4. The quantity of air passing is directly proportional to the square root of the head; the relationship between the air quantity, passing through the roadway at the given pressure, and R and A is simpler and clearer for the equivalent orifice; the quantity of air is directly proportional to the equivalent orifice and inversely proportional to the square root of the resistance; with increase of the equivalent orifice of the mine by a factor of 2, 3, or 5, the quantity of air passing at the same pressure is doubled, tripled or quintupled With reduction of the resistance in the same proportions, the air flow increases $\sqrt{2}$, $\sqrt{3}$, and $\sqrt{5}$ times.

Thus, although any change in the condition of the roadway (roughness of the walls, degree of dirtiness) and its dimensions is simpler and more clearly expressed as a change of resistance, the basic relationship of interest to us, that between the quantity of air passing through and the condition or dimensions of separate roadways or the mine as a whole, is simpler to express through the equivalent orifice

8-9. THROUGHPUT OF A MINE

The resistance R_m of any mine through which, at a pressure h mm, a quantity of Q m³/sec of air passes, is equal to $\frac{h}{Q^2}$. On the other hand, the quantity of air which will pass through this mine at a pressure of 1 mm is found from the equation

$$Q=\sqrt{\frac{h}{R_m}}=\frac{1}{\sqrt{R_m}} \tag{8-35}$$

This expression can be called the *throughput of the mine*. We shall call this value K, by analogy with the modulus of discharge or the throughput of a pipeline in hydraulics Because $R=\frac{h}{Q^2}$, then

$$K=\frac{1}{\sqrt{R_m}}=\frac{Q}{\sqrt{h}}\ \text{m}^3/\text{sec} \tag{8-36}$$

Comparing Expression (8-36) for the throughput K of the mine with Expression (8-30) for the equivalent orifice A, we see that Equation (8-30) differs from (8-36) by the coefficient 0 38 Although both indicate the degree of ease of ventilating the mine, their physical meaning is completely different the dimension of the equivalent orifice is area, the dimension of K is discharge

From Equation (8-36) it follows directly that the quantity of air passing through the mine at a head h is equal to

$$Q=K\sqrt{h} \tag{8-37}$$

It is therefore directly proportional to the throughput of the mine; this relationship is simpler than that of the air quantity to the resistance $\left(Q=\sqrt{\frac{h}{R}}\right)$ and to the equivalent orifice.

$$Q=\frac{A\sqrt{h}}{0.38}$$

It is not difficult to estimate that the throughput of a mine in relation to the classification given above will be equivalent to:

Type of mine	Value of K
Narrow mines	less than 2 64 m^3/sec per mm pressure
Medium mines	from 2 64 to 5 26 m^3/sec per mm pressure
Wide mines	more than 5 26 m^3/sec per mm pressure

8-10. THE CHARACTERISTIC OF A ROADWAY OR A MINE

The equation $h = RQ^2$ expressing the relation between the quantity of air flowing through a roadway or mine and the pressure loss in it is evidently a parabola. For a mine with a known resistance, it is drawn up in the following way. arbitrary values of the quantity of air Q_1, Q_2, Q_3, etc are substituted in the equation $h = RQ^2$, and the corresponding heads h_1, h_2, h_3, etc. are calculated. The calculations are made as follows:

Q	h
Q_1	h_1
Q_2	h_2
Q_3	h_3
etc.	etc.

On the axes h and Q of the co-ordinates are plotted points corresponding to the pairs of values $Q_1 - h_1$, $Q_2 - h_2$, and so on. These points are joined by a smooth curve which represents the equation $h = RQ^2$. This curve is called the *characteristic* of the roadway or the mine (Fig. 8-9).

The characteristics of all roadways pass through the origin of the graph and form a family of parabolas.

The characteristics of roadways with laminar flow of the air through them will evidently be straight lines passing through the origin.

The greater the resistance of the roadway or the mine, the steeper the characteristic will rise, and vice versa.

The characteristics of working mines or roadways are continuously changing. Changes in the air quantity flowing through them do not, however, change the characteristics, other things being equal

The actual characteristics of working mines are obtained in the following way in the shaft or incline near the fan drift a Pitot tube is placed, its static pressure end being connected to a water gauge at the surface; the air door is then gradually lowered, (or the diffuser is covered) and for each position the pressure h and the air quantity are measured. The pressures are corrected for the velocity pressure h_v at the point of measurement (see Chapter 7).

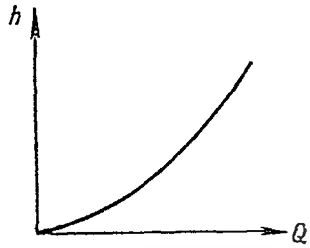

Fig 8-9 Graph of the characteristic of a mine or a district

The curve built up on the axes h and Q will be the mine characteristic. If the Pitot tube is a straight (static) pointed tube, it should be lowered into the shaft on a rubber tube slightly above the fan drift

The question arises whether the curve drawn up in this way really represents a parabola, as follows from the equation $h = RQ^2$, and whether it will pass through the origin The following answer must be given to these points

(a) if there is a natural draught, the curve will not pass through the origin, but will intersect the abscissa (Fig. 8-10), the intercept OA

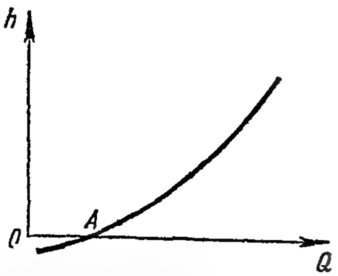

Fig 8-10 Graph of a mine characteristic in the presence of natural draught

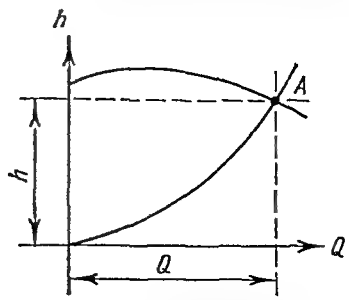

Fig 8-11 Curves plotted to determine the working conditions of a fan at a mine

corresponding to the amount of air entering the mine when the fan is stopped, owing to the natural draught;

(b) because the mine resistance, for the reasons indicated in Section 8-6, Subsections 1 and 3, may change somewhat with increase or reduction of the fan output, the characteristic is not necessarily a parabola.

Nevertheless, as will be seen in later chapters, a parabolic characteristic is widely used for graphical solutions of most varied problems in mine ventilation by fans or by natural draught; for this solution, apart from the characteristic of the mine, the *character-*

istic of the fan must also be given; this is a curve which expresses the relation between the quantity of air driven through the fan and the head developed by it.

If the characteristics of the mine and of the fan are known, and the quantity of air produced by the fan has to be determined for a mine with a particular characteristic, then evidently the required operating conditions of the fan will occur both on the mine characteristic (the curve joining the points of all possible ventilation conditions of the mine including the required one) and on the fan characteristic, which connects all the points of possible operating conditions of the fan, i.e. at the point A, the intersection between the fan characteristic and the mine characteristic; the ordinate of this point will be equal to the head h of the mine air, and the abscissa will be the air quantity Q passing through the fan (Fig. 8-11).

CHAPTER 9

TOTAL RESISTANCE OF THE MINE ROADWAY SYSTEM. THE AIR FLOW DISTRIBUTION

9-1. GENERAL

A *mine roadway system* may be any combination of mine workings, a whole mine or part of a mine. If air passes through the roads, it is a *ventilation system*

The *total resistance of any ventilation system* is a single value which represents in ventilation calculations the combined resistance of all the roadways and faces to the air passing through them; independently of the type of connection between the roadways and faces in the system, the total resistance is always less than (or equal to, with series connections of the roadways discussed below) the sum of the resistances of all the roadways in the system; thus

$$R_{total} \leqslant R_1 + R_2 + \ldots + R_n \qquad (9\text{-}1)$$

It is assumed (p 251) that the total resistance of the roadway system is a constant, independent of the air quantity passing through it

The solution of the many problems in mine ventilation planning generally reduces either to finding the total quantity of air which will pass through a given ventilation system with one or more fans of known dimensions and known rotational speed (rpm), or to finding the distribution of a known quantity of air through the workings in the system To solve these problems, we must find the resistance of the whole ventilation system

They are generally solved in the following sequence·

Stage 1 to find the total resistance of the combination of workings.

Stage 2. to determine the total quantity of air passing through the system.

Stage 3 to find the distribution of the air through the different workings.

In finding the total resistance of a given ventilation system consisting of a combination of several workings, the following problems can be met with:

(1) the head between the initial and final points of the system is known for an arbitrary quantity of air passing through it,

(2) the natural distribution of an arbitrary quantity of air through the system is known,

(3) the dimensions and structure of the supports in the workings are known, as well as the required distribution of air through them;

(4) only the dimensions of the airways are known, i e. their length, cross-sectional area, and type of support.

The last two problems are mainly encountered in ventilation planning; the first two are part of the operation of the mine.

9-2. DETERMINATION OF THE TOTAL RESISTANCE WHEN THE AIR DISTRIBUTION IS KNOWN

Case 1. If in a working mine the total quantity of air Q_t passing through the workings between two points A and B is measured and the pressure difference between these points is h_{AB}, then

$$R_t = \frac{h_{AB}}{Q_t^2} \tag{9-2}$$

in which R_t is the required total resistance of all the workings between points A and B, however complicated the connections between them may be.

Case 2. Let the system between points A and B consist of n air splits, each a combination of any number of separate airways; some of the airways may be shared between different splits, the air quantity Q_1, Q_2, . . , Q_n passing through each airway, or that passing through each of the splits from point A to point B, is measured. Then to determine the total resistance of the system it is sufficient to find the pressure drop between points A and B in any of the splits in the form of the sum

$$h_{AB} = R_1Q_1^2 + R_2Q_2^2 + \ldots + R_nQ_n^2 \tag{9-3}$$

in which R_1, R_2, . . , R_n are the known resistances of the different airways included in the split, and Q_1, Q_2, . . , Q_n are the measured quantities of air passing through them.

The total resistance R_t is again found from Equation (9-2). The validity of this law is derived from the statement that *the head of all the splits between any two points A and B will be equal, independently of their resistances, i.e. independently of their length and cross-sectional dimensions, and of the air quantity passing through them.* Therefore any split whatsoever will have the same depression h_{AB}.

Case 3. The dimensions of the airways are known as well as the air distribution through them. In this instance, unlike the previous one, the calculated heads of the separate splits included in the

system can be different. Therefore to calculate the total resistance of the system it is necessary to use Equation (8-13) for the pressure drop of all the splits between the initial (A) and final (B) points of the system, then to take the largest of these pressure drops, and then by the equation $R_t = \frac{h_{AB}}{Q_t^2}$ to determine the total resistance of the system.

Let us examine all three instances on examples

We are required to determine the total resistance of a ventilation system consisting of two faces, two haulage roads, and two return airways (Fig 9-1)

Case 1 The total air quantity passing into the mine is measured, $Q_m = 20$ m³/sec, and the mine pressure at the point F before the air stream passes into the fan drift, $h_m = 55$ mm

According to Equation (9-2) the total resistance of the mine is equal to

$$R_m = \frac{h_m}{Q_m^2} = \frac{55}{20^2} = 0.1375 \text{ kilomurg} = 137.5 \text{ murgs}$$

This method of determining the total resistance of a roadway system is the simplest and the most accurate. However, using this method in practice, one must remember that if the given underground system includes inclined and vertical roadways, which often happens, and the mean air temperature of the downcast (AB) and of the upcast (BDF and BCF) air currents are not the same, then the workings are subject to a draught pressure not measured by the water gauge connected at the point F (see Chapter 10) Consequently, to avoid obtaining understated values of the resistance, it is necessary to add the natural draught pressure h_n to the measured mine pressure h_m*; in this instance let it equal 5 mm, then the corrected mine pressure will be equal to

$$h_m^1 = h_m + h_n = 55 + 5 = 60 \text{ mm}$$

and the true value of the mine resistance is

$$R_{true} = \frac{60}{20^2} = 0.150 \text{ kilomurg}$$

When the measured mine pressure is not large, neglect of the natural draught pressure can lead to very serious errors in calculating the total mine resistance

Case 2 The total air quantity $Q_t = 20$ m³/sec by measurement, and the quantity of air passing through one of the circuits of the

* Alternatively the natural draught pressure may have to be subtracted if it opposes the work of the fan; methods of measuring and calculating the natural draught pressure are given in Chapter 10

system, e.g. *BDE* (Fig 9-1) $Q_1 = 8$ m³/sec; the dimensions and type of supports are known and consequently also the coefficient α of

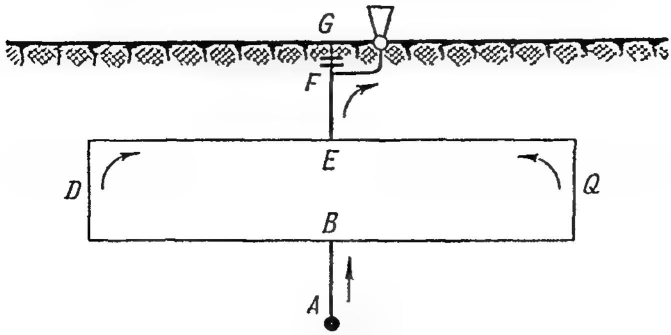

Fig. 9-1. Diagram for determining the overall resistance of the mine network

the airways through which this air passes, i.e

	L, m	P, m	S, m²	α	$R=\alpha\frac{LP}{S^3}$
Haulage road	2,200	10 2	6	0 0015	0 155
Face	100	8	4	0 0045	0 056
Return airway	2,200	8 32	4	0 0029	0 570

The pressure of the circuit *BDE* is equal to (0 155 + 0.056 + + 0.570)·8² = 50 mm; the total resistance R_t of the system *BDCE* is equal to

$$\frac{h_{BDE}}{Q_t^2}=\frac{50}{20^2}=0.125 \text{ kilomurg}$$

Case 3. The dimensions of all the airways of the system are given, as well as the air quantities flowing through the circuits: $Q_D =$ $= 10$ m³/sec and $Q_C = 8$ m³/sec. The dimensions of the airway *BDE* are the same as in the second case, the length of the airways in the split *BCE* is 880 m. The largest pressure in this instance will evidently be taken by *BDE* because it is longer

$$h_{BDE}=(0.155+0.56+0\ 570)\,10^2=78.1 \text{ mm}$$

and this will be the total pressure of the system; consequently the total resistance of the system *BDCE* will be

$$R_t=\frac{78\ 1}{18^2}=0.241 \text{ kilomurg}$$

Here we must note one extremely important point: because the resistance of the right-hand "wing" *BCE* is equal to 0.347 kilomurg

(calculations not given), then with a pressure difference between points B and E equal to 78 1 mm, in the right-hand "wing" we will have

$$Q=\sqrt{\frac{h}{R_{BCE}}}=\sqrt{\frac{78\ 1}{0\ 347}}=15\ \text{m}^3/\text{sec}$$

instead of the required 8 m³/sec

But the right-hand split is required to have only 8 m³/sec. To achieve this, it is possible, for example, to increase the resistance of the split up to $R=\frac{78\ 1}{8^2}=1\ 22$ kilomurgs, that is by $1\ 22 - 0.347 = 0\ 873$ kilomurg, which will correspond to an increase of the pressure by $h=RQ^2=0\ 873\times 8^2=55\ 9$ mm But since the pressure of the right-hand split with the original air quantity was equal to $0\ 347\times 8^2=22\ 2$ mm, the additional resistance required to be introduced in the right-hand split must be precisely such as to make its pressure equivalent to the difference in pressures of the two streams, i e $78\ 1-22\ 2=55\ 9$ mm.

Below some examples are considered showing how the total ventilating resistance of the system is found when only the cross-sectional dimensions, the type of support and length (case 4) of the system are known We shall look at series ventilation, parallel ventilation (splits), diagonal ventilation (compound networks), and a combination of these types of connection between workings

9-3. SERIES CONNECTION OF WORKINGS

If the ventilation system consists of several workings through which the air passes without branching out, it is called a *series-connected system*, or *coursing system* An example of series connection is the working of a single seam by one face (Fig 9-2) The air going down the shaft AB passes in series through the haulage cross-measure drift BC, the base horizon haulage road CD, the face DE, the return airway EF, the cross-measures drift FG, the upcast shaft GH, and the fan drift HK In practice, this sort of series ventilation of a single air flow through all the mine workings is exceptional; incomparably more often, the workings in series form only a part of a more complicated system.

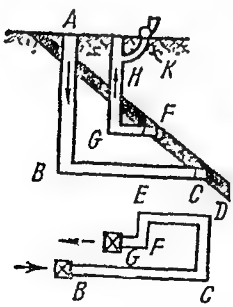

Fig 9-2 Example of airways in series

Let us denote the resistance of the separate workings forming the series system by R_1, R_2, . . . , R_n, etc It would appear that the total resistance of the series-connected system is the precise sum of their resistances

$$R_t=R_1+R_2+\dots+R_n \tag{9-4}$$

In reality this is not completely true. The fact is that in addition to the power spent on overcoming the frictional resistance R in each of the series-connected workings, additional power is spent on the redistribution of air flow rate as the air passes from an airway with one type of roughness and one cross-sectional area to another with a different roughness and cross-sectional area, except in such sudden enlargements and restrictions as will be mentioned below.

Usually this circumstance has little effect on the total resistance and is neglected; the total resistance of a series-connected system is therefore always taken as the sum of its separate resistances

Because $R = \frac{0.144}{A^2}$, then from Equation (9-4) we obtain the relation between the total equivalent orifice of several series-connected airways and their individual equivalent orifices

$$\frac{1}{A_t^2} = \frac{1}{A_1^2} + \frac{1}{A_2^2} + \ldots + \frac{1}{A_n^2} \tag{9-5}$$

Multiplying all the terms of Equation (9-5) by Q^2 (since $Q_1 = Q_2 = \ldots = Q_n$), we obtain

$$R_t Q^2 = R_1 Q_1^2 + R_2 Q_2^2 + \ldots + R_n Q_n^2$$

or

$$h_t = h_1 + h_2 + \ldots + h_n \tag{9-6}$$

that is, *the total pressure drop of series-connected workings is equal to the sum of their pressure drops.*

9-4. PARALLEL CONNECTIONS

Parallel connections between workings are those in which two or more workings branch out and rejoin later; they have no other air connection to each other apart from these two points.

A typical example of the simplest parallel connection is the case of two-wing working of one seam or ore deposit as shown in Fig. 9-3*a*

The air passing along the cross-measure drift AB divides at point B into two streams: BC and BE, which rejoin at point D to form a single stream DF; the streams BCD and BED are called *splits* Fig. 9-3*b* shows a simplified layout of two splits. It is quite obvious that two workings, though parallel from the point of view of ventilation, may not be geometrically parallel.

There is a distinction between *closed* and *open parallel* (or *split*) *systems*; closed split systems answer the description given above; open split systems, such as shown in Fig. 9-4*a* and *b*, are those in which the splits do not in fact rejoin at one point, but whose end

points are outlets from the mine to the atmosphere, the pressure drops of the splits are the same* and consequently they satisfy the main requirement for split systems.

The most important property of split systems is that *the pressure drops of splits are always equal and independent of their lengths, resistances, and the air quantities passing through them.*

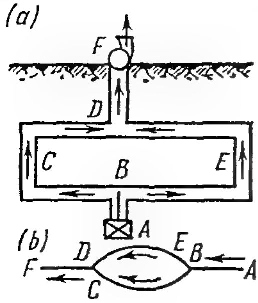

Fig 9-3 Example of a closed split system

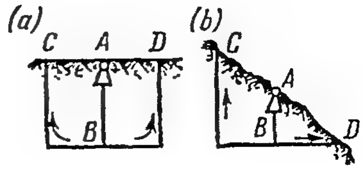

Fig 9-4 Example of an open split system

Thus, for example, the pressure drop between the atmosphere and the shaft before the fan drift (Fig 9-1) will be equal to (a) the total mine pressure h_{ABCEF}, that is the pressure of any split from the collar of the downcast shaft or shafts to the point of measurement of the pressure, as well as to (b) the pressure h_{GF} of the short length for the air sucked through the shaft building or the doors of the fan adit.

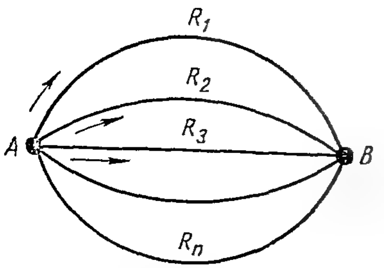

Fig 9-5 Diagram of several splits

This property of splits is uniformly characteristic of both closed and open splits, the pressures of splits ABC and ABD (Fig 9-4), as pointed out above, are equal. Only in the instance when each split, open or closed, is served by a different fan or a different natural draught pressure will the pressures in these splits be different.

The resistances of splits A system of several parallel workings (Fig 9-5) has the resistances R_1, R_2, . . , R_n and it is required to find the total resistance of the system.

Let us first find the total resistance of the two first splits Based on the rule for the equality of the pressures of splits we have

$$h_1 = R_1Q_1^2 = h_2 = R_2Q_2^2 = R_tQ^2 \qquad \text{(a)}$$

* Neglecting the difference in velocity pressures at the outlet to the atmosphere

from Equation (a) we obtain·

$$\frac{Q_1}{Q}=\sqrt{\frac{R_t}{R_1}} \tag{b}$$

and

$$\frac{Q_2}{Q}=\sqrt{\frac{R_t}{R_2}} \tag{b'}$$

Adding Equations (b) and (b′), we get:

$$\frac{Q_1+Q_2}{Q}=\sqrt{\frac{R_t}{R_1}}+\sqrt{\frac{R_t}{R_2}}$$

or after cancelling

$$\frac{1}{\sqrt{R_t}}=\frac{1}{\sqrt{R_1}}+\frac{1}{\sqrt{R_2}} \tag{9-7}$$

By analogy, for the entire system of workings, we obtain

$$\frac{1}{\sqrt{R_t}}=\frac{1}{\sqrt{R_1}}+\frac{1}{\sqrt{R_2}}+\quad+\frac{1}{\sqrt{R_n}} \tag{9-8}$$

Because $R=\frac{0\ 144}{A^2}$, we have $\frac{1}{\sqrt{R}}=\frac{A}{\sqrt{0\ 144}}$ and Equation (9-8) can be transformed as follows:

$$A_t=A_1+A_2+\ldots+A_n \tag{9-9}$$

It is used in this form when, instead of the resistances of separate workings, their equivalent orifices A are known.

For two parallel splits after simple transformation we obtain

$$R_t=\frac{R_1}{\left(\sqrt{\frac{R_1}{R_2}}+1\right)^2}=\frac{R_2}{\left(\sqrt{\frac{R_2}{R_1}}+1\right)^2} \tag{9-10}$$

For three splits we obtain·

$$R_t=\frac{R_1}{\left(\sqrt{\frac{R_1}{R_2}}+\sqrt{\frac{R_1}{R_3}}+1\right)^2}\text{ etc} \tag{9-11}$$

A comparison of Equations (9-7), (9-10), and (9-11) shows that the problem of finding the total resistance of splits is simpler to solve using the conception of the equivalent orifice When A_t is obtained, then

$$R_t=\frac{0\ 144}{A_t^2} \tag{9-12}$$

From Equations (9-10) and (9-11) we can see that the total resistance of a system of splits is always less than the resistance of each

split taken separately. This property of splits is widely used in practice, the air being, where possible, directed into several parallel roads The fan thus encounters less resistance and gives more air.

In the particular case when the resistance of two workings is the same, i e. when $R_1 = R_2$, we obtain from Equation (9-10)

$$R_t = \frac{R_1}{\left(\sqrt{\frac{R_1}{R_2}}+1\right)^2} = \frac{R_1}{4} \tag{9-13}$$

that is, the total resistance of two splits of the same resistance is only 25 per cent of their separate resistances

In the general form

$$R_t = \frac{R_{st}}{n^2} \tag{9-14}$$

in which n = the number of splits For laminar air flow we could obtain correspondingly

$$R_t = \frac{R_{st}}{n} \tag{9-15}$$

Equation (9-14) can be written in the form

$$R_{st} = R_t \cdot n^2 \tag{9-16}$$

enabling us to obtain the resistance of one of the splits when the total resistance of the system is known; e g if the resistance R_{reg} of a regulator of S m^2 is known, then, assuming that the regulator consists of S parallel elements having an area of 1 sq m each, the resistance of one element will evidently be equal to $R_{reg}S^2$ kilomurgs, by the quadratic equation

Let us denote the head of each of two splits by h_1 and h_2; then because the heads of the splits are equal, we will obtain:

$$h_1 = R_1Q_1^2 = h_2 = R_2Q_2^2 = h$$

from which we have

$$Q_1 = \sqrt{\frac{h}{R_1}} \quad \text{and} \quad Q_2 = \sqrt{\frac{h}{R_2}}$$

in which h = the total identical head of the splits. Adding these two equations we obtain

$$Q_1 + Q_2 = Q_t = \sqrt{\frac{h}{R_1}} + \sqrt{\frac{h}{R_2}} \tag{9-17}$$

or

$$h = \frac{Q_t^2}{\left(\sqrt{\frac{1}{R_1}}+\sqrt{\frac{1}{R_2}}\right)^2} = R_tQ_t^2 \tag{9-18}$$

We have already used Equation (9-18) in order to obtain the total resistance R_t of a ventilation system from measured values of h and Q_t.

Considering the particular example of the use of the equation for determining the total resistance of splits, let the total quantity of air emerging from the mine (Fig. 9-6) be Q_m, and that passing by leakage from the surface through a door be $p\%$; the resistance of the mine is R_m and that of the mine and the door together, R_t.

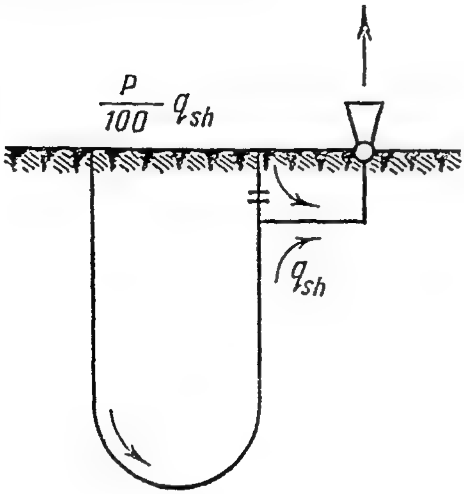

Fig 9-6. Diagram showing method of determining the pressure drop between the atmosphere and the shaft, upstream of the fan drift

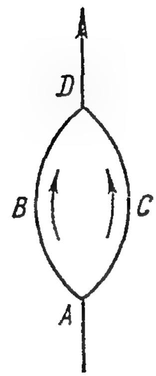

Fig 9-7. Diagram for calculating a split system

It is required to find R_t. The solution is as follows since the air quantity sucked in through the door is evidently $\frac{p}{100}\cdot Q_m$, the resistance of the door R_d is obtained from the proportions

$$\frac{R_d}{R_m}=\frac{Q_m^2}{\left(\frac{p}{100}Q_m\right)^2}$$

from which we obtain

$$R_d=\left(\frac{100}{p}\right)^2\cdot R_m$$

Consequently the total resistance from Equation (9-10) is equal to

$$R_t=\frac{R_m}{\left(\sqrt{\frac{R_m}{\left(\frac{100}{p}\right)^2\cdot p_m}}+1\right)^2}=\frac{R_m}{\left(\frac{p}{100}+1\right)^2}=\lambda R_m \qquad (9\text{-}19)$$

Substituting in this formula, instead of p, the various percentages of air sucked in through the regulator, we obtain the following values for the leakage factor k, equivalent to $\frac{1}{\left(\frac{p}{100}+1\right)^2}$:

Leakage, per cent	Leakage factor, k	Leakage, per cent	Leakage factor, k
5	0.907	30	0.597
10	0.825	40	0.510
15	0.755	50	0.445
20	0.694	60	0.396
			etc.

This table will be extremely useful in the analysis of the work of fans in Chapter 11.

The distribution of the air. Let there be two splits ABD and ACD (Fig. 9-7), they will have the same pressure drop

$$h_{ABD}=h_{ACD} \quad \text{or} \quad R_1Q_1^2=R_2Q_2^2 \tag{a}$$

in which R_1 and R_2, Q_2, and Q_1 are the resistances and the air quantities in each split. From Equation (a) we obtain

$$\sqrt{\frac{R_1}{R_2}}=\frac{Q_2}{Q_1}$$

that is, *the ratio of the air quantities is inversely proportional to the square root of the ratio of the resistances.*

Adding a unity to each part of this equation, we obtain

$$\sqrt{\frac{R_1}{R_2}}+1=\frac{Q_2}{Q_1}+1=\frac{Q_2+Q_1}{Q_1}=\frac{Q_t}{Q_1}$$

hence

$$Q_1=\frac{Q_t}{\sqrt{\frac{R_1}{R_2}}+1} \tag{9-20}$$

In the same way

$$Q_2=\frac{Q_t}{\sqrt{\frac{R_2}{R_1}}+1}$$

For n splits we therefore obtain

$$Q_1=\frac{Q_t}{\sqrt{\frac{R_1}{R_2}}+\sqrt{\frac{R_1}{R_3}}+\ \dots\ +1} \tag{9-21}$$

Let, for example, the resistances R_1 and R_2 of two splits be equal to 1.0 and 0.5 kilomurg, respectively, then the ratio

$R_1 \cdot R_2 = 2$, from Equation (9-20) the air quantity flowing into these splits will be:

$$Q_1 = \frac{Q_t}{\sqrt{\frac{1\ 0}{0\ 5}}+1} = 0.415Q_t$$

and

$$Q_2 = \frac{Q_t}{\sqrt{\frac{0\ 5}{1\ 0}}+1} = 0\ 585Q_t$$

The ratio $Q_2/Q_1 = 0\ 585/0\ 415 = 1\ 41 = \sqrt{2}$.

If the equivalent orifices A_1 and A_2 of the different splits are known, as well as the equivalent orifice A of the whole system, it is easy to show that

$$Q_1 = \frac{Q_t A_1}{A_t} \tag{9-22}$$

That is, *the air is distributed between the splits in direct proportion to their equivalent orifices*

9-5. DIAGONAL CONNECTION OF SPLITS

A diagonal connection of two splits is one in which they are diagonally joined together at some point between their start and their finish, by one or more additional intermediate workings (Fig. 9-8*a* and *b*).

The workings BC (Fig. 9-8*a*) and BC, DE and FG (Fig 9-8*b*) are called "*diagonals*". A network with one diagonal is called a *compound network of the first order*; with two or more diagonals it would be a *compound network of the second* or *third order*. The main feature of all diagonal connections is the presence of one or more workings through which the air can travel in opposite directions or not at all, depending on the resistances in the other roads

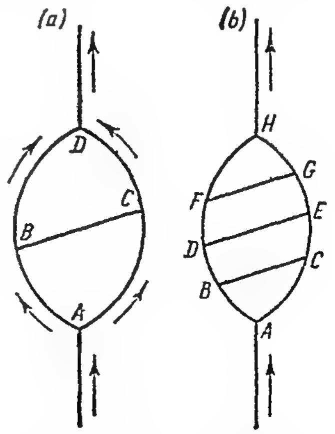

Fig 9-8 Diagrams of compound networks

For example, in the compound network of the first order shown in Fig. 9-8*a* the direction of air flow through the roadway sections AB and BD, AC and CD will always be from point A to points B and C and from them to point D; reverse movement of the air is evidently unthinkable. However, along the diagonal BC the air can move either from B to C or from C to B. Indeed, if the resistance of the

section BD by comparison with the total resistance of the two sections BC and CD is considerable, then the air from point B can in part flow along the "easier" path BC. On the other hand, if the section BD has a small resistance, most of the air in the network will pass through it to point D avoiding the "difficult" section CD, in this instance the air will pass along the diagonal from C to B.

It is quite obvious that the presence in a system of underground roadways of such sections in which the air can move in opposite directions requires exceptional vigilance on the part of the ventilation staff, and strict observance of the rules for keeping ventilation

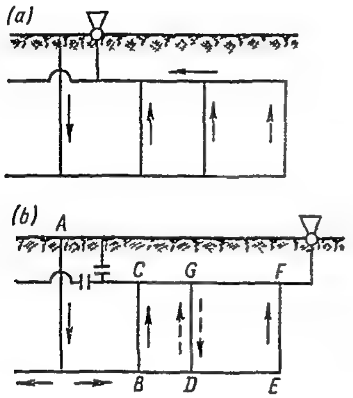

Fig 9-9 Example of compound network underground

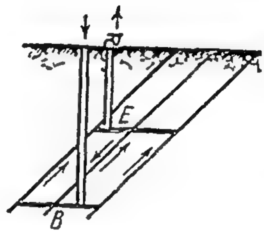

Fig 9-10 Compound network in the mining of seams close together

doors shut The accidental opening of any door or closing of another can change the relationship between the various resistances so that the air flow in the diagonals will reverse

In practice, diagonal connections are frequently found underground Fig 9-9a with the upcast and downcast shafts in the centre of the mine take shows a system of simple splitting; the air can travel only in the direction shown by the arrows, if, however, the upcast shaft is transferred to the boundary of the take (Fig. 9-9b), the ventilation layout becomes a compound network, with a diagonal DG in which the air can move in either direction Let us imagine, for example, that because of an explosion a fall has taken place in the district, partly blocking the cross section of some working, the air movement along the diagonal would be reversed and the poisonous products of the explosion could unexpectedly appear in the section GDE with disastrous consequences

Another example of a compound network is shown in Fig. 9-10, where the diagonal evidently is the middle seam BE

Fig. 9-11, finally, shows the ventilation layout often used for working gassy seams by long faces retreating, the air does not all pass through the whole face, but is gradually added by the intermediate roadways CB and DE; the diagonal of this compound network will evidently be CB.

These few examples do not exhaust the possible types of diagonal connections underground.

The first graphical solution of a diagonal connection was given in 1908 by Professor G. O. Chechett in Russia. Professor A. S. Popov and others later worked out simpler analytical approximate solutions, and solved the problem of the compound network of higher

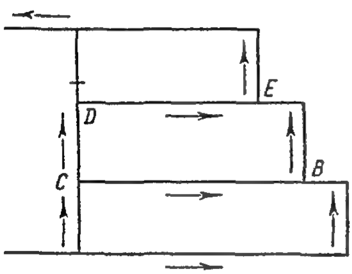

Fig. 9-11 Compound network as exemplified by a retreating method in a gassy mine

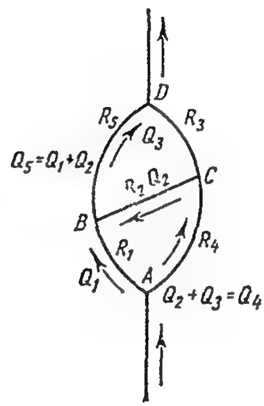

Fig. 9-12. Diagram for calculating a compound network

order. Finally, very recently Professor F. A. Abramov suggested a new and more accurate analytical method.

The solution of the diagonal connection problem includes: (a) determining the direction of the air flow through the diagonal; (b) determining the total resistance of the network; (c) determining the air distribution through the network.

Let us assume that we have a compound network of the first order as illustrated in Fig. 9-12. Let the resistance of the separate sections AB, BC, CD, AC and BD of this network be equal to R_1, R_2, R_3, R_4 and R_5, and the air flow through them be Q_1, Q_2, Q_3, Q_4 and Q_5.

Evidently, there will be a certain ratio between the resistances R_1, R_2, R_3, R_4 and R_5, at which there will be no air flow through the diagonal BC. Let us find this ratio.

No air will flow through the roadway BC when the pressure at the points B and C is the same; for this to occur, h_{AB} should be equal to h_{AC} ($h_{AB} = h_{AC}$), and similarly $h_{BD} = h_{CD}$; dividing the first equation by the second and substituting RQ^2 for the pressure drops.

we obtain, after eliminating Q^2

$$\frac{R_1}{R_5}=\frac{R_4}{R_3} \tag{9-23}$$

This evidently is the ratio between the resistances of the sections in a compound network, at which no air passes through the diagonal

If we increase or reduce the resistance of any working of the compound network, except the diagonal, the air flow will be redistributed, and air will pass through the diagonal Thus, for example, an increase of the resistance of the roadways CD or AB will cause air to flow through the diagonal in the direction from C to B, Equation (9-23) in this case will be changed to the inequality

$$\frac{R_1}{R_5}>\frac{R_4}{R_3} \tag{9-24}$$

which will also be the necessary and sufficient condition for the movement of air along the diagonal from point C to point B

The analogous inequality

$$\frac{R_1}{R_5}<\frac{R_4}{R_3} \tag{9-25}$$

is a sufficient condition for the flow of air along the diagonal from point B to point C.

Since none of the Equations (9-23), (9-24) and (9-25) contains any term of the resistance of the diagonal itself, we can rightly conclude that the *resistance of the diagonal itself has no effect on the direction of air in it.*

Let us now find the air distribution in a compound network of the first order The total air flow Q through the system is known, and so are the resistances R_1, R_2, R_3, R_4 and R_5 of the separate roadways forming the network (Fig 9-12) It is required to find the air quantities flowing through all the roadways of the network

Let us assume that the air flows along the diagonal from C to B, and denote the quantities of air flowing through the sections AB, BC, and CD, by Q_1, Q_2, Q_3, respectively, section AC receives an air flow Q_2+Q_3 and section BD receives the quantity Q_1+Q_2

The pressure heads of the air flows AB and ACB (Fig 9-8) should be the same

$$h_{AB}=h_{AC}+h_{CB}$$

or

$$R_1Q_1^2=R_4(Q_2+Q_3)^2+R_2Q_2^2 \tag{a}$$

In the same way the pressure heads of the flows CD and CBD are the same

$$h_{CD}=h_{BC}+h_{BD}$$

or

$$R_3Q_3^2=R_2Q_2^2+R_5(Q_1+Q_2)^2 \tag{b}$$

Dividing Equations (a) and (b) by Q_2^2, we obtain

$$R_1\left(\frac{Q_1}{Q_2}\right)^2 = R_4\left(1+\frac{Q_3}{Q_2}\right)^2 + R_2 \qquad \text{(c)}$$

$$R_3\left(\frac{Q_3}{Q_2}\right)^2 = R_2 + R_5\left(\frac{Q_1}{Q_2}+1\right)^2 \qquad \text{(d)}$$

Let us denote the ratios $\frac{Q_1}{Q_2}$ and $\frac{Q_3}{Q_2}$ by x and y, respectively, then the Equations (c) and (d) will become:

$$R_1x^2 = R_4(1+y)^2 + R_2 \qquad \text{(c')}$$

$$R_3y^2 = R_2 + R_5(x+1)^2 \qquad \text{(d')}$$

The analytical solution of these two equations of the second degree with two unknowns is laborious Let us therefore use the simpler, although less accurate, graphical solution to find the roots of the equations

The equations are in fact two hyperbolas:

$$\frac{x^2}{\frac{R_2}{R_1}} - \frac{(y+1)^2}{\frac{R_2}{R_4}} = 1 \text{ and } \frac{y^2}{\frac{R_2}{R_3}} - \frac{(x+1)^2}{\frac{R_2}{R_5}} = 1$$

The co-ordinates of the points of intersection of these two hyperbolas are evidently the required roots x, y of Equations (c) and (d). Generally speaking, hyperbolas can intersect in all four quadrants, and consequently the values of x and y can be either positive or negative. However, since x and y are ratios of air quantities flowing through the roadways, they can only be positive, the problem therefore has a single, positive solution, determined by the intersection of the hyperbolas in the first quadrant The necessary and sufficient condition for this single positive solution will therefore be the inequality

$$\frac{R_1}{R_5} > \frac{R_4}{R_3}$$

which indicates that the air flows from C to B If the air flow is in the reverse direction, from B to C, it is necessary to change R_1 for R_4, and R_5 for R_3 after which the problem is solved in a similar way.

To draw the hyperbolas from Equations (c) and (d), we determine x and y.

$$x = \sqrt{\frac{R_4(1+y)^2 + R_2}{R_1}} \qquad \text{(e)}$$

$$y = \sqrt{\frac{R_5(1+x)^2 + R_2}{R_3}} \qquad \text{(f)}$$

By substituting several arbitrary values for x and y in Equations (e) and (f) we obtain a number of paired values of x and y. Each pair of values of x and y, of which the first is arbitrarily selected, and the second is calculated from Equations (e) and (f), consists of the co-ordinates of points on the hyperbolas, joining these points by smooth curves we obtain the required sectors of the hyperbolas.

The hyperbolas are conveniently plotted on log paper. The point of intersection of the hyperbolas gives the required roots x and y of Equations (c') and (d').

To solve the problem of the air distribution along the roadways in a compound network we use the following relations

$$x = \frac{Q_1}{Q_2} \quad \text{and} \quad y = \frac{Q_3}{Q_2}$$

Let us then write the identity $1 = \frac{Q_2}{Q_2}$ and add the three equations

$$x + y + 1 = \frac{Q_1}{Q_2} + \frac{Q_2}{Q_2} + \frac{Q_3}{Q_2} = \frac{Q}{Q_2}$$

from which

$$Q_2 = \frac{Q}{x + y + 1}$$

and consequently

$$Q_1 = xQ_2 = \frac{xQ}{x + y + 1} \quad \text{and} \quad Q_3 = yQ_2 = \frac{yQ}{x + y + 1} \tag{9-26}$$

Equations (9-26) solve the problem of the air flow distribution through the roadways of a compound network.

The pressure drop of a compound network, i.e. the pressure difference between the points A and D (Fig. 9-12) is the sum of the pressure drops of any air flow between the two points.

Thus, for the flow AB we obtain

$$h_{AD} = h_{AB} + h_{BD} = R_1 Q_1^2 + R_3 (Q_1 - Q_2)^2$$

or, after substituting the values of Q_1 and Q_2

$$h_{AD} = \frac{x^2 R_1 + R_3 (1 - x)^2}{(x + y + 1)^2} Q^2 \tag{9-27}$$

Similarly it would be possible to obtain three more equations for calculating the total depression of a compound network by taking the air flows $ACBD$, $ABCD$, or ACD.

The expression standing in front of Q^2 in Equation (9-27) represents the total resistance R_t of the whole compound network. The question arises, is this resistance larger for the same values of R in a simple split system ABD (ACD) or is it smaller? In other words,

what is the effect of the diagonal BC on the resistance of a split system? The answer to this question can be obtained from the following consideration if the ends of the diagonal are joined to points B and C in the splits such that $\frac{AB}{BD} = \frac{AC}{CD}$, the air does not flow through the diagonal, and the presence of the diagonal will have no effect on the resistance of the network; if the points B and C are moved towards A and D or D and A, the system will tend to resemble one with three splits, in which the resistance is always less than in one with two similar splits. For any other position of the diagonal the total resistance of the system will be equal to some intermediate value between a two-split system and a three-split system, thus, it will always be less than the resistance without a diagonal.

Compound networks including those of higher order can nowadays be solved by electric models (Section 9-7).

9-6. COMBINED CONNECTION OF AIRWAYS

Let us look at a combination of the various underground airways. Nearly all mine ventilation layouts include both series, and parallel districts, and sometimes also compound networks In the simplest instances the total resistance of this combination of airways is obtained by gradually eliminating all the parallel and compound networks and substituting their total resistances to which are then added the series sections until the total resistance of the system is obtained Sometimes a simplification is used in which some roadways of small resistance are ignored.

9-7. ELECTRIC MODELLING

Modelling of mine workings, i.e the building of models to scale in the laboratory, for example, to obtain their aerodynamic resistance, is often used. Such models are called aerodynamic models If they model a shaft, they are circular in cross section, if they model a roadway, the cross section will be rectangular or trapezoidal. The scale of the model (M) is usually from 1/5 for large models to 1/10 for small ones; Dr. P. I Mustel has successfully used models at a scale of 1/100.

Sometimes a model of several workings is made, with a turning from one working into another or the junction of the pit bottom with the shaft, or the shaft with the fan drift, etc

Modelling of the whole circuit of the mine is almost never used because of its complexity and the difficulty of maintaining the value of the Reynolds number at a level which ensures constant aerodyna-

mic resistance At the same time, there is the need for models of various types for investigating ventilation systems. The generally known graphic-analytic method of solving ventilation layouts is fairly cumbersome and complicated and does not solve all the problems in mine ventilation planning nor in the analysis of the ventilation of a working mine.

The electric modelling method is therefore exceedingly helpful It is based on the analogy which can exist between the equations referring to a particular physical process and other equations referring to the distribution of electric currents and voltages in a specially designed electric circuit. The method is not new; it was used as long ago as 1918 by N P. Pavlovsky for investigating the seepage of water However, it was not until 1952 that the first description of the method in mining was published in the German mining magazine *Gluckauf*

The use of electrical models enables solutions to be found for complex and sometimes impossibly difficult ventilation problems by the direct measurement of electric currents and voltages, with the subsequent conversion of these values to the aerodynamic values of flows and pressures, using appropriate scales.

The flow of mine air in the most general case is described by the following equations

$$\left.\begin{aligned} h &= R'_a Q \\ h &= R_a Q^2 \\ h &= f(Q) \\ \Sigma Q_i &= 0 \\ \Sigma h_i &= 0 \end{aligned}\right\} \qquad (9\text{-}28)$$

in which R'_a = the aerodynamic resistance of an element of the ventilation circuit with laminar air flow, $\frac{\text{kg sec}}{\text{m}^5}$

R_a = the aerodynamic resistance of an element of the ventilation circuit with turbulent flow, $\frac{\text{kg sec}^2}{\text{m}^8}$

$f(Q)$ = a function indicating the characteristic of the fan

ΣQ_i = the algebraic sum of the air flows directed to any node of the network and leaving this node

Σh_i = the algebraic sum of the pressure drops in the closed circuit

The last equation in system (9-28) is called the law of single-valued pressures at the point In writing the equations according to this law, one of the directions of air flow (e g clockwise) is taken as positive, the other (e g. anti-clockwise) being negative, if the

air flow in the closed circuit is in one direction, the sum of the heads will include the fan head with a sign opposite to that of the pressure loss in the circuit.

The direct-current electric circuit is described by a system of equations which is fully analogous to system (9-28):

$$\left.\begin{aligned} V &= R_0' I \\ V &= R_0 I^2 \\ V &= f(I) \\ \Sigma I_i &= 0 \\ \Sigma V_i &= 0 \end{aligned}\right\} \qquad (9\text{-}29)$$

in which V = tension, volts
I = current, amperes
$f(I)$ = a function of the current source
R_0' = the linear electrical resistance, ohms
R_0 = the quadratic electrical resistance, V/A^2.

The last two equations in system (9-29) express the first and second of Kirchhoff's laws which are applicable to any electric circuit.

The resistances used in the electrical model can be of various types, e.g.:

metal conductors,
ordinary incandescent lamps,
miniature incandescent lamps,
rheostats,
radio valves and semi-conductors

To convert the electrical values obtained in the model to the aerodynamic values, the following relationships are used:

$$\left.\begin{aligned} h &= m_h \cdot V \\ Q &= m_Q I \\ R_a' &= m_{R_a'} \cdot R_e' \\ R_a &= m_{R_a} \cdot R_e \end{aligned}\right\} \qquad (9\text{-}30)$$

in which m_h, m_Q, $m_{R_a'}$, and m_{R_a} are the scales of pressure drop, air flow, and linear and quadratic aerodynamic resistances.

These scales are interrelated in a definite manner and therefore cannot all be chosen arbitrarily. This relationship is easily established if we divide the first two equations of system (9-28) by the

first two equations of system (9-29).

$$\frac{h}{V}=\frac{R'_a}{R'_e}\ \frac{Q}{I'}$$

$$\frac{h}{V}=\frac{R_a}{R_e}\ \left(\frac{Q}{I}\right)^2$$

or, including Equations (9-30)

$$m_h=m_{R'_a}\cdot m_Q \tag{9-31}$$

$$m_h=m_{Ra}\ m_Q^2 \tag{9-32}$$

From Equations (9-31) and (9-32) one further equation follows, linking the scales together

$$m_{R'_a}=m_{Ra}\ m_Q \tag{9-33}$$

Since two independent Equations (9-32) and (9-33) link four values m_h, m_Q, $m_{R'_a}$, m_{R_a}, only two scales can be chosen arbitrarily in the electric modelling of mine ventilation circuits, and the remaining two must be determined by Equations (9-31), (9-32) or (9-33)

The dimensions of these scales are as follows:

$$m_h=\frac{\text{kg/m}^2}{\text{volts}}$$

$$m_Q=\frac{\text{m}^3\text{/sec}}{\text{amp}}$$

$$m_{R'_a}=\frac{\text{kg sec/m}^5}{\text{ohm}}$$

$$m_{Ra}=\frac{\text{kg sec}^2\text{/m}^3}{\text{volt/amp}^2}$$

The use of ordinary incandescent lamps for electric modelling in turbulent flow underground is discussed below In this case we have, from Equations (9-28) and (9-29)

$$h=R_a\ Q^2 \quad \text{and} \quad V=R_0\ I^2$$

But the quadratic relation between current and voltage, analogous to the quadratic relation between air flow and depression can be observed only when we replace the value R_0 (ohmic resistance) by the value $\frac{R_0}{I}$;

then

$$V=\frac{R_0}{I}\cdot I^2=R_0 I=kI^2 \tag{9-34}$$

in which $k=\frac{R_0}{I}$.

This indicates that the aerodynamic resistance R_a corresponds in an electrical model to the value $k = \frac{R_0}{I}$ (the so-called proportionality factor), the air flow corresponding to the value of the electric current I, and the pressure to the voltage V

Equation (9-34) shows that the ohmic resistance of the electrical model should vary in direct proportion to the value of the current

$$R_0 = kI$$

or, which is the same, that the ratio $k = \frac{R_0}{I}$ should be constant.

Investigations into the use of ordinary incandescent lamps at the laboratory of the Mining Institute at Dnepropetrovsk have shown that it is possible to choose from a large number of tested lamps those which satisfy the conditions $V = kI^2$ and $k = \frac{R_0}{I} = \text{const}$ within a certain voltage interval

For the lamps investigated, the value of the actual relation $k_{act} = \frac{R_0}{I}$ ohms per amp was shown for various groups of lamps to be between 990 and 845, with deviations not exceeding 3-5 per cent within the groups From these lamps connected in series, in parallel, or in series-parallel, it is possible to obtain resistances of various values For the series connection of n lamps, the total resistance is equal to nk_{act}, for the parallel connection of n lamps, it is $\frac{k_{act}}{n^2}$ For modelling it is evidently necessary to accept some ratio between the electrical resistance and the aerodynamic resistance, a conversion factor, e g we may arbitrarily assume that a lamp with $k_{act} = 882$ corresponds to 1 kilomurg Then two lamps connected in series will have a resistance of 2 kilomurgs Bearing in mind that the smallest number of lamps is the most convenient, it appears that a more suitable method for aerodynamic modelling might be to use some arbitrary conversion factor other than that used here, $k_{act} = 882$. A simple example of a problem solved by this method is given below

Example. The mine workings shown in Fig 9-13 consist of sections with resistance $R_1 = 23$, and $R_2 = 25$, and two splits with resistances $R_3 = 100$, and $R_4 = 115$, the total air flow through the system is 24 m³/sec It is required to find how the air is distributed through the workings

Solution. *Stage 1* Let the available lamps have an actual proportionality factor determined experimentally and equal to $k_{act} = \frac{R_0}{I} = 882$ ohms per amp

By trial and error we choose a suitable arbitrary value of aerodynamic resistance, appropriately related to the value k_{act} which yield all the required resistances from a minimum number of lamps by connecting them in series, parallel,

and series-parallel Thus, assuming that one lamp with k_{act} = 882 ohms per amp has a resistance of 48 murgs, we find that 1 murg corresponds to $\frac{882}{48}$ = 18 35 ohms/A Then with a series connection of n such lamps their total resistance will be n 48, and with parallel connections it will be $48 \cdot n^2$

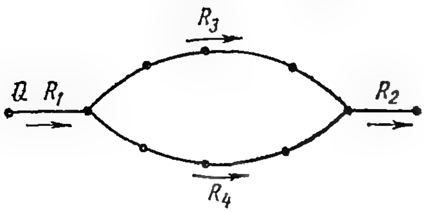

Fig 9-13 A split layout of airways

Fig 9-14 shows how, by various combinations of lamps, the necessary resistance values in the different sections can be obtained, e g the resistances R_1 and R_2, equal to 23 and 25 murgs, are obtained by series connections of two groups of parallel connected lamps, giving k_1 and $k_2 = \frac{48}{2^2} + \frac{48}{2^2} = 24$ with an error of $\frac{24-23}{23}$ 100 = 4 3 per cent

Let us now recall once more that the relation $\frac{k_{act}}{R} = \frac{\text{ohms/A}}{\text{murg}}$ can be chosen arbitrarily, but the given resistances R_1, R_2, , R_n will then be in each instance built up from various combinations of lamps

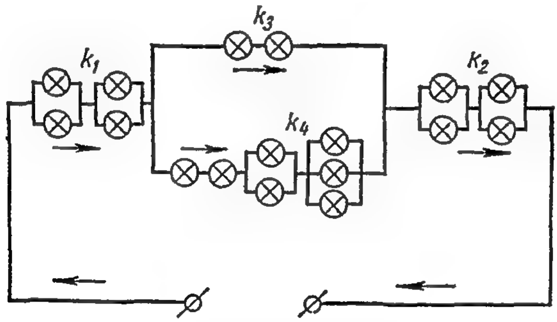

Fig 9-14 Circuit diagram of an electrical model of a split system

For example, arbitrarily taking the value of 882 ohms per amp to be not 48 but 72 murgs, we obtain, with parallel connection of 2, 3 and 4 lamps, $\frac{R}{2^2} = \frac{72}{4} = 18$, $\frac{R}{3^2} = \frac{72}{9} = 8$, $\frac{R}{4^2} = \frac{72}{16} = 4\ 5$ Consequently a resistance $R = 23$ murgs can be obtained from a layout with one group of two lamps in parallel, series-connected to one group of four lamps in parallel $R_1 = 18 + 4\ 5 = 22\ 5$, with an error of $\frac{23 - 22\ 5}{23} \times 100 = 2\ 17$ per cent The resistance $R_3 = 100$ murgs can be built up thus $72 + \frac{72}{4} + \frac{72}{9} = 98$, and so on

Stage 2 The electrical diagram of the model is drawn to the ventilation layout shown in Fig 9-14 and the model is finally erected

Stage 3 The voltage is fed into the circuit and recorded with a voltmeter, $V = 564$ V, and an ammeter, $I_t = 0\ 625$ A

Stage 4. We determine the scales (conversion factors) for converting from current to air flow, m_Q, and from electrical voltage to pressure m_h

(a) $m_Q = \frac{Q}{I} = \frac{24}{0.625} = 38.4$ and $m_Q' = \frac{I}{Q} = 0.026$ A $= 26$ mA, that is, 26 mA in the electrical circuit corresponds to 1 m³/sec in the air flow,

(b) to determine m_h we find first of all the total resistance of the circuit, from Equation (9-34) we obtain $k = \frac{V_t}{I_t^2} = \frac{564}{0.625^2}$ ohm/A and since 18.35 ohms/A corresponds to 1 murg (see above) then $R_t = \frac{564}{0.625^2 \times 18.35} = 76$ murgs

Finding the total resistance, we determine the total pressure of the circuit

$$h = \frac{R_t Q_t^2}{1{,}000} = \frac{76 \cdot 24^2}{1{,}000} = 43 \text{ mm}$$

consequently $m_h = \frac{h}{V_t} = \frac{43}{564} = 0.076$ and $m_h' = \frac{V_t}{h} = \frac{564}{43} = 13.1$, that is, 13.1 V on the electrical model corresponds to a pressure of 1 mm water gauge

Stage 5 We measure the current and the voltage in all sections of the model and with the help of the conversion factors ($m_Q' = 26$, and $m_h' = 13.1$) we convert them to the values of air flow and pressure This conversion is done in Table 9-1 which also compares the values of Q and h obtained by calculation from the usual formulas

TABLE 9-1 **Conversion of Air Quantities and Pressures**

Section	I, mA	Air flow, Q m³/sec			Voltage, V	Pressure, h mm H_2O		
		model	calculated	deviation		model	calculated	deviation, per cent
	625	24	24	—	564	43	43	
r_1	625	24	24	—	176	13.4	13.3	0.75
r_2	625	24	24	—	178	13.6	13.8	1.45
r_3	325	12.5	12.5	—	147	14.7	15.2	3.28
r_4	300	11.5	11.5	—	147	14.7	15.2	3.28

Note When the lamps in the set differ from each other in their K value by only 5 per cent, the layout can be chosen without the preliminary selection of lamps

9-7.1 The Electrical Model of the Mining Institute of the Soviet Academy of Sciences, Used for Designing Mine Ventilation Networks

Recently, various electrical models have been proposed for determining the air flow distribution in underground networks These models differ from each other in the analogy for the resistance to air flow underground. In the USSR in 1952 V. N. Voronin and

A. D. Bagrinovsky suggested the use of incandescent lamps for this purpose, but because of a number of disadvantages these models were not widely used. In 1956 F. A. Abramov and N. A. Frolov proposed to use rheostats controlled by L. F. Moshnin's follow-up systems. In 1957 A. D. Bagrinovsky and R. V. Zubov worked out several electronic circuits. Simultaneously F. A. Abramov and V. A. Boiko proposed to use transistors for this purpose. Investigations by A. D. Bagrinovsky showed that the most advanced, simple, and easily adapted electrical models of an underground network were based on the use of linear-piecewise approximation circuits fed with direct current.

Fig. 9-15 shows a general view of the electrical mine network calculator of the Soviet Academy of Sciences, in which the underground circuits are modelled on the linear-piecewise approximation principle. The installation includes 64 models of underground workings whose resistances can be varied between 4 and 400 murgs. The required values of resistance are set by hand controls, operated from the switchboard. The installation has 4 power sources reproducing the head characteristics of fans. Each power source has an ammeter showing the flow from the fan, and a voltmeter showing the head created by the model of the fan. In the lower part of the installation there are jacks for the measuring instruments, and a switching device enabling the models of workings and fans to be connected to each other in any required way. The installations are supplied from an a-c network through voltage stabilizers. The method of calculation of the ventilation networks on the installation is as follows. Connecting the models of the workings and fans in the same way as they would be joined underground, the power sources are arranged so that their characteristics agree with the head characteristics of the appropriate fan. The models of mine workings are set with the required values of the resistance factors and the current is switched on. Then, using the ammeters and cathode-type voltmeters, the current and voltage distributions in the electric model are measured, giving the distribution of air flow and head in the ventilation circuit.

The results of a model investigation on the compound network shown in Fig. 9-12 were obtained from the following known values of resistances $R_4 = 0.12$, $R_1 = 0.16$, $R_3 = 0.21$, $R_5 = 0.114$, $R_2 = 0.07$ kilomurg, and the total air flow in network 30 m³/sec. We assume the following conversion factors.

$$m_k = \frac{R}{k} = 0.001 \frac{\text{kg sec}^2 \text{ mA}^2}{\text{m}^8 \text{ V}}$$

$$m_Q = \frac{Q}{I} = 50 \frac{\text{m}^3}{\text{sec mA}}$$

$$m_H = \frac{H}{V} = 2.5 \frac{\text{kg}}{\text{m}^2 \text{ V}}$$

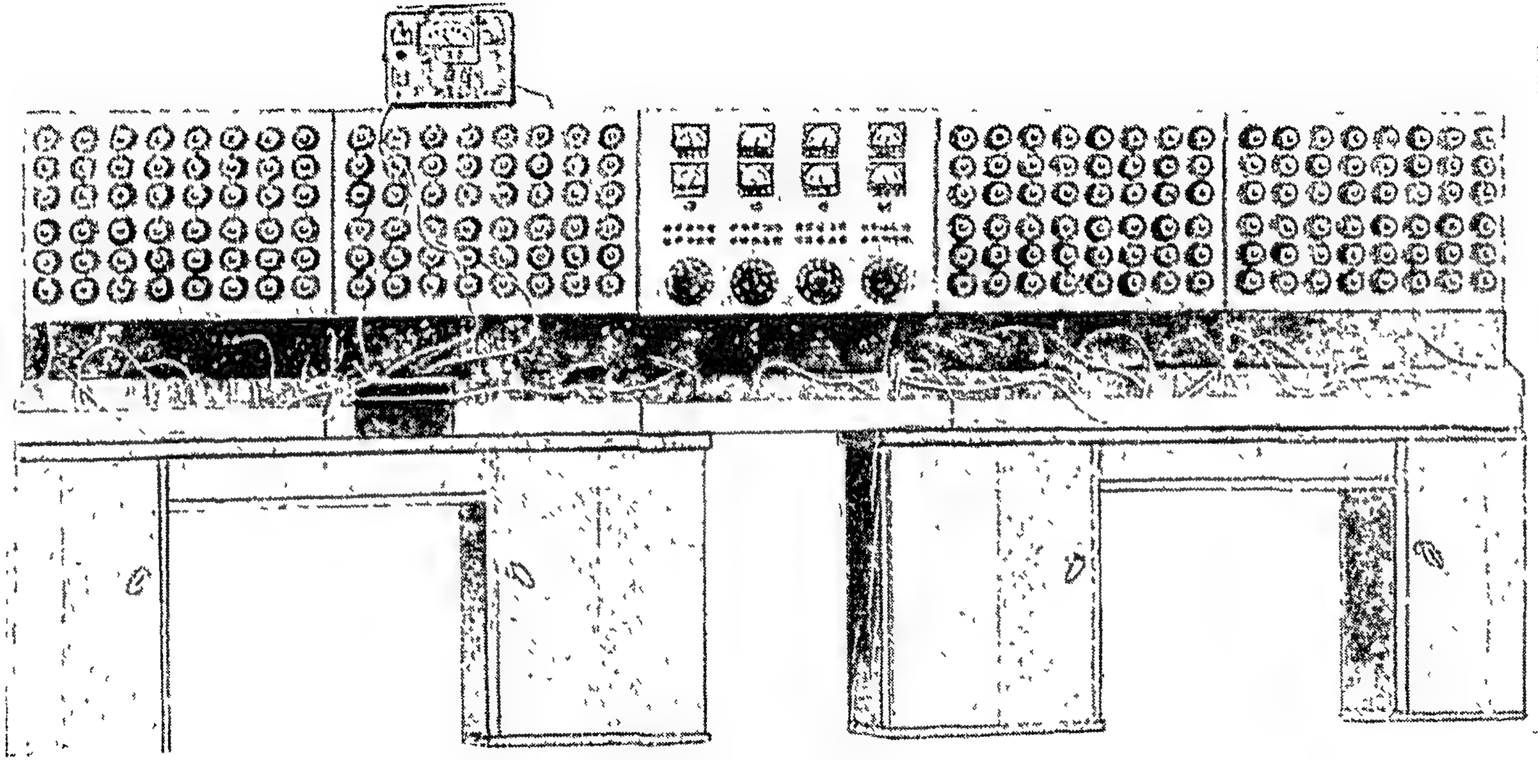

Fig 9-15 General view of an experimental instrument for calculating mine ventilation networks, designed by the Mining Institute of the USSR Academy of Sciences

Consequently we should establish the following resistances in the analogues of the workings $k_4 = 120$, $k_1 = 160$, $k_3 = 210$, $k_5 = 114$, $k_2 = 70$ The total value of the current in the connections will be

$$I_t = \frac{30}{50} = 0\ 6\ \text{mA}$$

The model is connected and installed with a current supply of 0 6 mA The total voltage drop in the model will be 25 6 V corresponding to a head in the mine of $H = 2\ 5 \times 25\ 6 = 64$ kg/m² We then measure the current in all branches of the circuit and convert these values to air flows The results are indicated in the table below in which there is also a comparison with the distribution in a compound network calculated analytically

Roadway No	Current, mA	Air flow in model, m³/sec	Calculated, m³/sec	Deviation, m³/sec	Deviation, %
4	0 32	16	16	0 00	0.00
1	0.275	13 75	14	0 25	1 7
3	0 245	12 3	12.8	0 5	3 9
5	0 35	17 5	17 2	0 3	1 7
2	0 065	3 25	3 2	0.05	1 56

The actual total head will be 64 8 mm H_2O, consequently the deviation in head is 1 mm, or 1 6 per cent From the data given above we can conclude that this model enables the air distribution in an underground circuit to be calculated with a maximum error of 4 per cent

The installation is built up from separate units, each containing up to 20 roadways; from several such units it is possible to build up a model of a more complex network One of the advantages of the model is its relative cheapness, one element of a model airway costing about a tenth of an automatic rheostat controlled by a follow-up system The model is also smaller in size The parameters of these systems are practically invariable with time The whole model is built up from standard radio components, and the computation of underground networks takes much less time than in models with hand-operated rheostats

9-8. CONCLUSION

To conclude this chapter we must point out the following.

In determining the total resistance of any roadway in a working mine, the results obtained will be reliable only to the same degree as the measurements of air flow Q and head h When the air flow is measured, one must be confident that the quantity obtained is not accidental, but is the actual air flow normally circulating

through the underground system; therefore the measurements must not be limited to one or two, even if the results are in agreement, and many measurements must be made for at least 10 to 15 minutes. The same applies to measurements of head; in each individual instance it is necessary to decide whether the natural draught should be taken into account. All measurements of Q and h are better made in a non-working shift, and preferably simultaneously.

For determining the total resistance of a system of workings during planning, reliable values of coefficients of friction must be used. In addition we must remember that the cross sections of mine workings do not remain constant (this mainly concerns timbered workings) and consequently the planned (calculated) values of resistances very soon cease to correspond to the actual condition of the mine. However, it is very difficult to give a general solution to this problem. We can observe that, other conditions being equal, a reduction of cross-sectional area by 5 per cent involves a 13 per cent increase of resistance; a 10 per cent reduction of area involves 26 per cent more resistance; a 15 per cent reduction of area involves 41 per cent more resistance; and a 20 per cent reduction of area results in 58 per cent more resistance.

As an example of the increase of resistance of workings with age, we can mention investigations made in Georgian mines (USSR): some 2 to 3 years after the roads had been driven (main haulage roads, and inclines) their resistance had increased by about 40 per cent, and the resistance of roadways in advancing longwall workings had even increased by 70 per cent; the roads driven in the rock (cross-measures drifts and main haulage roads) suffered an increase of resistance of 15 per cent. The rock, containing coal seams, was weak.

In areas of steady rock pressure, the cross-sectional areas of the workings remain practically constant. Workings supported with steel or reinforced concrete supports also have small variations in cross-sectional area. Finally, in some mines the workings are systematically repaired to their original design dimensions

One further point should be noted

In the analytical solution of problems of finding the total resistance of a system of airways, air losses are generally ignored, and the resistance calculated is known to be too high. In mine planning it is possible to set oneself a value of losses, using fairly reliable standards of air losses, such as those indicated in Chapter 14 (hitherto these standards have only been established for the conditions of the Donets Basin). In operating conditions, the locations of the losses, their magnitudes, and sometimes also the pressure drops they cause can be determined by ventilation surveys. The leakages of air passing through the ventilation doors, walls, and the wastes

are then shown on the ordinary ventilation plan. Some of these air courses may be arbitrary, since the direction of air seepages cannot always be found accurately Diagrams drawn up with an allowance for these losses are fairly complicated and often cannot be worked out by usual methods laid down in ventilation courses In this case great help can be given by the electrical network calculators described briefly in Section 9-7, they enable the total resistance of the most complicated ventilation layouts to be determined This method is also of value because it enables the effect of the natural ventilating pressure of the heated air to be estimated.

CHAPTER 10

THE NATURAL DRAUGHT

10-1. GENERAL

One of the features of mine ventilation is that usually the air can flow through the workings without any mechanical help.

To distinguish it from artificial draught by a fan or any other ventilating device, this air flow is called *natural draught.*

Since, as pointed out in Chapter 7, air can only flow when there is a pressure difference, by the natural draught one should understand the natural pressure drop in mine workings.

The dimension of this draught is kg/m^2 The natural draught (Chapter 7) is numerically equal to the work done by 1 cu m of air on its entire course through the mine. This statement also applies to the head developed by the fan

The natural draught in mines results from: (a) two or more vertical or inclined shafts;* (b) air of different densities in these shafts Difference in density alone is not enough to create a natural draught in the absence of vertical or inclined columns of air.**

To refer to Fig. 10-1 as an example, in winter, cold and consequently heavy outside air in the column *AB*, together with *BC*, will force out of the mine the lighter, warm column of air *DE*, the column *BC* will be the downcast shaft. The great Russian scientist M. V. Lomonosov noted this effect as early as 1742 in his thesis "On the free movement of air observed in mines' and worked out the first theory of natural draught. He wrote that in winter the air should pass into the lower shaft *BC* and pass out of the upper shaft *DE*.

The main cause of the varying specific weight of air is temperature difference; to a smaller degree there is the effect of variation in pressure caused by the fan, and to an even smaller degree the difference in humidity and chemical composition of the air.

Natural draught occurs in the following way.

---

* In shafts, during the sinking work, natural draught in summer sometimes consists of air passing down the walls and up the centre of the shaft, and in winter in the reverse direction

** We neglect the small movement of air which takes place when air is heated or cooled in horizontal workings, since this movement is the consequence of expansion or contraction of the air volume

1 The shaft collars are at the same level, in this case there is no reason for any temperature difference between the two shafts, and therefore natural draught is absent or very small. However, mines do exist in which, with this location of the shafts, there is a natural draught for different reasons, e g in a wet shaft the air is always cooler than in a dry one and the air will be drawn down.

2 During its work, the fan sucks cold air into the mine in winter, and warm air in summer, as a result, the natural draught in winter is in the same direction as the fan draught, whereas in summer it usually opposes the fan draught, the value of this natural draught may reach 20 to 25 per cent of the fan head.

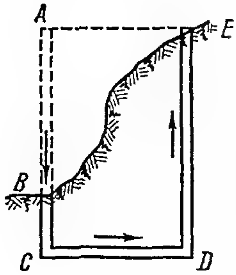

Fig 10-1 Diagram of the initiation of natural draught, with shaft collars at different altitudes

3 When a mine on the slope of a hill is worked through an adit, the natural draught results from the temperature difference between the air in the mine and that outside; in winter the adit is the intake, and in summer it is the outlet for the air In autumn and spring when the temperature of the air underground and outside is more or less the same, there is no natural draught and the ventilation deteriorates

Natural draught occurs also with an underground fire when the hot products of combustion fill the inclines and vertical shafts; this draught very often causes serious disturbance to the ventilation of the mine; a natural ventilation head which opposes the fan head is particularly dangerous.

The natural draught is extremely important for mine ventilation. In small mines, particularly those working non-fuel minerals, the natural draught is even now often the only means of ventilating the mine In deep mines with artificial ventilation the natural draught affects the work of the fan, either helping it or opposing it.

In mines of medium depth, (400 to 500 m) the value of the natural draught reaches 25 to 30 mm, in deep mines (700 to 800 m) it reaches 50 to 60 mm, and in very deep mines (1,000 to 1,200 m) it may exceed 120 mm. When the fan is stopped in cold weather the natural draught may occasionally reach 40 to 50 per cent of the air flow ordinarily produced by the fan A particularly large air flow from the natural draught occurs in mines with large equivalent orifices Thus, at some Michigan copper mines which are 1,500 to 1,600 m deep, the natural draught yields 4,000 m^3/min through one mine At the Kirov apatite mine in the USSR the natural draught alone, according to measurements by a team of the Leningrad Mining Institute, gave 6,000 m^3/min on a hot summer day Many more examples could be cited It can be seen therefore

that the natural draught is very important in deep mines with a large temperature difference between the downcast and the upcast air Natural draught acting in the same direction as the air flow, caused by the fan, can be considered as a safety factor If, however, the natural draught opposes the fan, it is essential to consider it in the ventilation planning of the mine. Frequently, neglect of this point in ventilation design may lead to gross errors.

10-2. MEASUREMENT OF THE NATURAL DRAUGHT

Assume a mine with a natural draught; we stop the fan, open the doors of the building above the upcast shaft, and try to measure the natural draught by the use of a water gauge, one arm of which is connected to a rubber tube lowered down the open shaft The water gauge will indicate no pressure; this result could have been foreseen, because the pressure on the water on both arms of the water gauge is the same, and this is obvious, subject to the strict condition that the air temperature in the rubber tube is the same as that of the outside air.

Consequently a *water gauge connected to the open top of the upcast shaft will not measure the natural draught*

We will try another experiment We close the doors of the upcast shaft and connect the water gauge to a Pitot tube connected to the fan drift, if the fan is stopped and the natural draught is positive, the water gauge will measure the small head spent in overcoming the resistance of the air passing through the fan; this head is so small that an ordinary water gauge will not measure it; therefore we can state that in this instance, too, the natural draught forcing the air through the mine cannot be measured, obviously the same will happen when the fan starts to work again; a water gauge installed on the surface and connected to the fan drift does not measure it

It is, however, possible to measure the natural draught. Below are given some methods of measuring h_n

1. A gate is lowered into the fan drift, or the whole ventilating current in the mine is cut off by interposing doors at some point; the pressure drop across this door and wall will be equal to the natural draught; the measurement of h_n should be made as quickly as possible after the fan has stopped so that the air temperature does not have time to change.

2 The upcast shaft is opened, the fan is reversed and the upcast shaft doors are gradually closed until the air flow in the upcast shaft stops completely; at this instant the pressure difference between the space in the shaft under the half-closed doors, and the outside air will be equal to the natural draught.

The accuracy of methods 1 and 2 is affected by the presence of several working horizons, since local natural draught circuits are formed between them

3 In spring or autumn, when there is no natural draught, the following method can be used the resistance of the mine $R = \frac{h_f}{Q_f^2}$ is determined, the mine characteristic is drawn to this resistance (Fig 10-2), then in a period when the natural draught is operating again, the air quantity Q_{f+n} is measured, and the intercept AB will be equal to the natural draught. A disadvantage of the method is that the value of the resistance R may slightly change in this period.

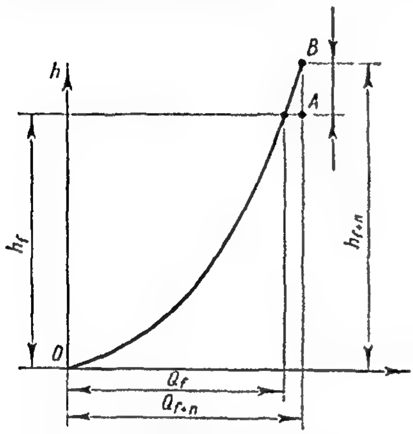

Fig 10-2 Diagram for determining the natural ventilating pressure

4 The following is also a simple way of determining h_n: with the fan working, the air flow Q_m passing through the mine, and the pressure h_m are measured by one of the usual methods The fan is then stopped, a gate is lowered into the fan drift, and the doors of the building over the upcast shaft are opened; after several minutes, when the air flow has become steady, the quantity of air, Q_n, flowing through the doors, is measured (at low air flow rates, only one leaf of the door need be opened)

From the two equations $h_m + h_n = R_m Q_m^2$ and $h_n = R_m Q_n^2$, it is easy to find both h_n and R_m

Methods 1 and 4 were checked at mines in the Donets Basin by S K. Kilkeyev and yielded concordant results

The following remarks should also be made on the subject of natural draught. When natural draught exists, and any measurement of pressure drop in any mine working is made, the natural draught pressure at this point is also measured, whether or not the fan is working On this basis, pressure surveys can also be made in mines with purely natural ventilation However, the natural draught is not measured by the pressure difference across the bulkhead of an underground booster fan either when the fan is working or when it is stopped, when the fan is stopped the pressure difference measured can only be the loss through the fan To measure the natural draught, the fan must be completely closed or a bulkhead must be built right across the working at a point through which the whole air flow of the mine passes, as in method 1.

10-3. CALCULATION OF THE NATURAL DRAUGHT

Since the value of the natural draught is equal to the difference in the specific pressures of two columns of air of different density, it cannot be calculated without knowing the temperature at the top and the bottom of each of these columns of air. Generally speaking, the column of air from the top of the shaft down to the lowest horizon takes part in forming the natural draught. We can assume for simplicity that the downcast shaft extends to the lowest working horizon. The necessary air temperatures are measured in the following way.

The air temperature t_1 at the top of the downcast shaft is usually known from the given conditions these are the minimum (February) or the maximum (July) air temperature for the given locality (obtainable from an observatory or meteorological office), if the air is heated, the winter temperature can be taken as $+2°C$. It is more correct to use, as the intake temperature t_1', the air temperature at 30 to 50 m from the surface· in this case the average temperature $\frac{t_1'+t_2}{2}$ is approximately equal to the average temperature in the shaft (with an error of ± 1 per cent) calculated on the basis of 15 measurements at various depths.

The temperature t_2 at the bottom of the downcast shaft can be found from the following equation proposed by the Academician A N. Shcherban

$$t_2 = -19.6 + \sqrt{A + \frac{H}{3.42}} \tag{10-1}$$

in which H = the depth of the shaft in metres.

Numerical values of the coefficient A for the Donets Basin mines are given in Table 10-1.

TABLE 10-1 Values of A for the Donets Basin Mines

Month	A	Month	A	Month	A
January	432	May	1,187	September	1,187
February	486	June	1,392	October	907
March	648	July	1 470	November	648
April	907	August	1,392	December	486

As a first approximation, Equation (10-1) can also be used in other coalfields

To determine the temperature t_3 at the bottom of the face, i e to determine the increase of temperature of the air as it passes through the horizontal roadways, A N. Shcherban has proposed various fairly complicated formulas. For an approximate estimate of the natural draught, we can assume that the air temperature at the foot of the upcast shaft is equal to the original rock temperature (which is known if the geothermic gradient is known) less 2-3°C in summer and less 4-5°C in winter When the temperature at the foot of the upcast shaft is determined, with two central shafts, and a connection drift between them, the in-leakage of cold air must be taken into account, which reduces the upcast air temperature by several degrees, particularly in winter The air temperature will be the higher for greater lengths of underground air streams and for lower air flow rates A N. Shcherban considers that with an underground air stream of about 1 to 3 5 km, the seasonal variations in temperature even at depths of 1,000 to 1,200 m will reach 5 to 7°C in the face

The temperature t_4 at the top of the upcast shaft depends on the degree of heating of the air in the production faces, which is affected by the oxidation processes, and the heat exchange between the rock and the air stream In particular, in coal mines this exchange reaches 3 to 7°C In copper pyrite mines, the increase of air temperature in the working stopes may exceed 7 to 10°C, depending on the efficiency of the ventilation. Within the upcast shaft, we can say approximately that its air temperature diminishes by 0 4°C for every 100 m upwards.

Equations for calculating the natural draught Before deducing the formulas, we must point out that the natural draught is affected by: (a) the movement of the air underground; (b) the work of the fan.

It is known that the pressure of the air column moving towards the foot of the shaft is less, and the pressure of the air column moving towards the top of the shaft is greater than the pressure of the still air column in the shaft Consequently, the natural draught with moving air will be somewhat less than with still air in the shafts. The difference will increase with shaft depths and aerodynamic resistances; it can be taken roughly that for depths of about 1,000 m, neglecting the air movement leads to the understatement of the natural draught by several millimetres

The effect of the work of the fan is not so clear On the one hand, it would seem that as a result of the head created by an exhaust fan, the specific weight of the air in the upcast shaft should reduce (and with a forcing fan would increase in the downcast shaft), which in both instances should slightly increase the natural draught.

On the other hand, A. V. Voropayev has shown that the value of the natural draught calculated without and with the effect of the fan usually does not differ by more than 1 to 2 per cent, rarely by 3 to 4 per cent. The problem cannot be considered settled and requires further study.

Below are given some "static" formulas for calculating the natural draught. The values can be obtained from the equation

$$h_n = p_1 - p_2 \tag{10-2}$$

in which instead of p_1 and p_2 we substitute the values of the specific pressures due to the air column in the downcast and upcast shafts

$$\gamma'_{av}H \quad \text{and} \quad \gamma''_{av}H$$

in which γ'_{av} and γ''_{av} are the average specific weights of the air in each of the shafts, they are calculated by Equation (4-30) from the average air temperatures t'_{av} and t''_{av} for which the arithmetic means of the temperatures at the top and the bottom of the shaft can be used as a first approximation

Substituting these specific weights in Equation (10-2) we obtain

$$h_n = (\gamma'_{av} - \gamma''_{av})\ H = 0.455 p_0 \left(\frac{1}{273 + t''_{av}} - \frac{1}{273 - t'_{av}}\right) \tag{10-3}$$

in which p_0 is the barometric pressure whose approximate values are obtainable from the table on p. 192 Another formula for evaluating h_n can be deduced from Equation (6-4) for the isothermic law, in which p_0 = the air pressure at the top of the shafts.

$$p_1 = p_0 \cdot e^{\frac{H}{RT'_{av}}} \quad \text{and} \quad p_2 = p_0 \cdot e^{\frac{H}{RT''_{av}}}$$

Consequently

$$h_n = p_0 \cdot e^{\frac{H}{RT'_{av}}} - p_0 \cdot e^{\frac{H}{RT''_{av}}} \tag{a}$$

Expanding both expressions $e^{\frac{H}{RT_{av}}}$ in Equation (a) and rejecting the members beginning with the third and multiplying the right-hand part by 13.6, we obtain

$$h_n = p_0 \frac{H}{100}\left(\frac{13.6\ 100}{RT'_{av}} - \frac{13.6\ 100}{RT''_{av}}\right) =$$

$$= p_0 \cdot \frac{H}{100}\ (a_1 - a_2) \text{ mm water gauge} \tag{10-4}$$

The values of a_1 and a_2 for various temperatures can be taken from the following table.

t_{av} °C	—30	—25	—20	—15	—10	—5	0	5	10	15	20
a	0 191	0 187	0 184	0 180	0 177	0 174	0.170	0 167	0 164	0 161	0.159

For values of H larger than 100 m, the value of h_n must be multiplied by the correction factor $\left(1 + \frac{H}{10{,}000}\right)$.

Temperature measurements in a downcast shaft of a mine in the Donets Basin showed that in winter the air temperature increases by 5-7°C for the first 50 m, therefore in calculating the value of T'_{av} in the Donets coalfield the temperature t_1 of the intake air in winter (when heated) should be increased by 5 to 7°C, below 50 m the temperature gradient should be taken as approximately 0.4 to 0 5°C for every 100 m depth in summer and up to 1°C per 100 m depth in winter

Another simple expression for the rapid calculation of h_n for $p = 760$ mm is

$$h_n = a \cdot \frac{H}{100}(t''_{av} - t'_{av}) \text{ mm } H_2O \tag{10-5}$$

in which t'_{av} and t''_{av} are the mean temperatures of the downcast and upcast air and the factor a is taken from the table below.

Values of the factor a

Depth of shaft, m	Temperatures of downcast and upcast air			
	$t'_{av} = 0$ $t''_{av} = 10$	$t'_{av} = 5$ $t''_{av} = 15$	$t'_{av} = 10$ $t''_{av} = 20$	$t'_{av} = 15$ $t''_{av} = 20$
100	0 47	0 45	0 43	0.42
400	0 49	0.47	0 45	0 43
1,000	0 52	0 51	0 49	0 47

Example Determine h_n if $p_0 = 750$ mm, $t'_{av} = 5$°C, $t''_{av} = 20$°C, and $H = 500$ m

Solution. From the table above, for a we obtain $a_1 = 0\ 167$, $a_2 = 0.159$, consequently from Equation (10-4) we have

$$h_n = 750 \cdot \frac{500}{100}(0\ 167 - 0.159) = 30 \text{ mm}$$

Correction factor is equal to

$$1 + \frac{500}{10{,}000} = 1.05$$

Therefore finally

$$h_n = 30 \times 1.05 = 31.5 \text{ mm}$$

By the less accurate Equation (10-5) we have

$$h_n = a \frac{H}{100}(t''_{av} - t'_{av}) = 0\ 46 \frac{500}{100}(20-5) = 34.5 \text{ mm}$$

There is one further simple formula which gives fairly accurate results

$$h_n = 0.0047H\,(t''_{av} - t'_{av}) \qquad (10\text{-}6)$$

in which t'_{av} and t''_{av} = mean air temperatures of the downcast and upcast shafts

H = the depth of the shaft, metres.

The calculation of h_n during the operation of the fan. A. F. Voropayev has worked out the following graphical method of determining the value of the natural draught On the axes of co-ordinates H and t, where H = the depth, m, and t = the temperature, degrees C, points of measured temperatures throughout the air flow are plotted from the collar of one shaft to the collar of the other shaft (Fig 10-3).

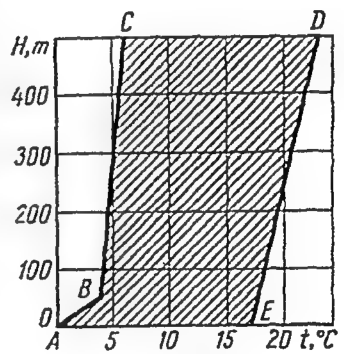

Fig 10-3 Graphical method of determining the natural ventilating pressure

The area S (the dimension of S is metres °C) within the closed surface $ABCDE$ is measured by a planimeter. The natural draught is equal to

$$h_n = \frac{S}{T_{av}}\gamma \text{ mm } H_2O \qquad (10\text{-}7)$$

in which T_{av} = the average absolute temperature of the centre of gravity of the figure, or as a first approximation the average absolute temperature of the two extremes, i.e. the points A and D;

γ = 1.2 kg/m³.

When the fan is working, all the points of the right-hand branch, beginning with the point E, should be lowered by the height h_m/γ in which h_m is the mine pressure measured at the top of the upcast shaft. The resulting area S' enclosed below by the horizontal line is deducted from the area S.

Fig. 10-3 is plotted for the conditions of the example above.

The hatched area $S = 7,650$ m°C; the average of the extreme absolute temperatures at points A and D is equal to

$$T_{av} = \frac{273+296}{2} = 284\ 50° \text{C}$$

$$h_n = \frac{7,650}{284\ 5} \times 1.2 = 32\ 3 \text{ mm } H_2O$$

Thus all three equations give fairly close results.

A comparison of the pressures calculated from Equations (10-4), (10-5) and (10-6) with pressures directly measured by water gauge in the ventilation laboratory of the Leningrad Mining Institute and checked at mines in the Donets Basin by Sh Kh Kilkeyev has shown that the smallest divergence between the calculated and measured values is given by Equation (10-4), the largest by (10-6)

Natural draught of the mine as a whole. When work is simultaneously carried on at several different horizons, the natural draughts calculated for separate streams will be different What should be taken for the natural draught of the mine as a whole?

Let us call Q_t the total air flow; let Q_1, Q_2 and Q_3 be the quantities of air in the separate splits, and h'_n, h''_n, h^i_n, the corresponding natural draughts The work done by the air will be equal to the sum of the work done in every split $\sum Q_i\ h^i_n$ which equals $Q_t\ h_n$, in which h_n is the required total pressure, therefore

$$h_n = \frac{Q_1 h'_n + Q_2 h''_n + \ldots + Q_i h^{(i)}_n}{Q_t} \tag{10-8}$$

10-4. FACTORS AFFECTING THE VALUE OF THE NATURAL DRAUGHT

(a) *The change of chemical composition and humidity of the air.* These changes are so small that the differences they cause in specific weights of air almost never affect the natural draught, and therefore they are usually neglected in the calculation of h_n; they could only be taken into account in mines which are exclusively naturally ventilated

(b) *Change in barometric pressure* From Equation (10-4) it follows that the value h_n is directly proportional to the *barometric pressure* p_0, but because the fluctuations of p_0 do not usually exceed ± 25 mm Hg, which is about 3 per cent of the normal atmospheric pressure, the effect of changes in p_0 on the value of h_n is also small

(c) *The effect of depth of the mine* As a first approximation, it can be assumed that the value of h_n is directly proportional to the depth H of the mine

(d) *The effect of the fan head* This effect was discussed above The fan head, as pointed out, has little effect on the value of h_n.

(e) *The effect of the air temperature* For a mine of given depth, the air temperature in the vertical shafts and inclines is the main factor affecting the value of h_n, but the temperature of the upcast air, as proved by observations at mines, changes very little in the

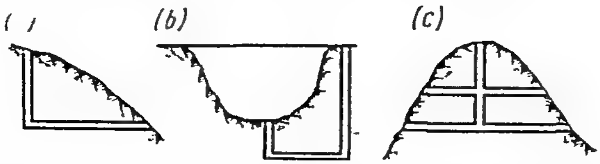

Fig. 10-4. Diagram to show how natural draught starts in the mine at various seasons and times of the day

course of the year, consequently *the main factor must be the temperature of the downcast air which depends on the season of the year* And indeed, we sometimes observe sharp fluctuations in the value of h_n in the course of the year, while in shallow mines which are opera-

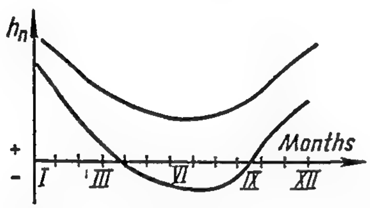

Fig. 10-5 Change in the level of the natural ventilating pressure through the year

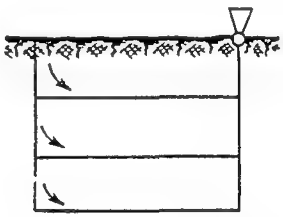

Fig 10-6 Boundary ventilation used in the simultaneous mining of several horizons

ted by adits or form a deep quarry connected with underground workings, here may even be changes in the direction of h_n in autumn and spring (Fig 10-4*a*, *b*).

In summer in areas with a continental climate, i e. with cool nights, there are also daily changes in the direction of the air flow at nights the quarry or adit is the intake airway, by day it is the return airway. Finally if the mine is located on a mountain side which is cut through by tunnels (Fig. 10-4*c*), the sun, heating the east slope of the mountain in the morning and the west slope in the evening, causes the mine air to flow from one side in the morning and from another in the evening

The annual changes of the absolute value, and sometimes also the direction, of the natural draught are shown on the diagram of Fig 10-5

These changes in direction of the natural draught refer to simple ventilation layouts with two shafts and one connecting drift between them, and to the layouts shown in Fig. 10-4. In reality, with a branching mine network and different air temperatures in the branches, there are more than one natural draught in the mine and they act like several fans working simultaneously; this will be discussed below Let us here point out merely the following features.

1. In deep mines with several simultaneously working horizons and a ventilation layout with a shaft at the boundary (Fig 10-6), when the downcast air temperature changes, there is either a stagnation or a reversal of the air current in the upper horizons, while the air in the lower horizons still flows in the same direction.

2. Sometimes, with a high resistance in the intake airways and producing faces, and conversely with a small resistance in the faces, a mine subject to varying temperatures in the shafts may have fairly powerful air circulations, considerably exceeding the intake of air from the surface

3 In workings to the dip or to the rise, which do not communicate directly with the return horizon, the temperature difference between the intake and the return air stream will generally help in the ventilation of the dip, and hinder the ventilation of the rise workings.

4 In underground fires, the heating of the air current removing the hot gases may rapidly reverse the air flow

5. Sometimes, in a mine with a fan, a desired change in the direction of the natural draught can be achieved by starting the fan and forcing the cold or hot air for a time through some vertical shafts in the required direction, the fan is then stopped, but the same direction of air flow is maintained

In a coal mine with continuously running fans, fluctuations in the value of the natural draught and changes in its direction are taken into account in the ventilation planning stage (Part Three) and therefore do not interrupt the ventilation The situation is quite different in metal ore mines where fans are sometimes only used after blasting, and then stopped The usual consequences of these stoppages, particularly in broken country, is the freezing up of the shaft collars or adits in winter and the reversal of the ventilating current in the production faces in summer These disturbances are difficult to avoid and usually only the changeover to continuous operation of the fan completely eliminates them.

10-5. THE CHARACTERISTIC OF THE NATURAL DRAUGHT

The characteristic of the natural draught is a curve which relates the value h_n to the air flow passing through the mine. In practice this curve can be plotted by somehow varying the air flow through

the mine with a fan, and measuring the natural draught or calculating it from the temperature measurement. By this method, in winter, the increased intake of cold air due to the increased fan throughput will increase the natural draught, while in summer it will diminish.

Thus the theoretical characteristic of h_n should be obtained in the form of curves rising in winter and falling in summer with increase of Q (Fig. 10-7).

In practice, to simplify ventilation calculations, since they are in any case relatively crude in mine conditions, the characteristic

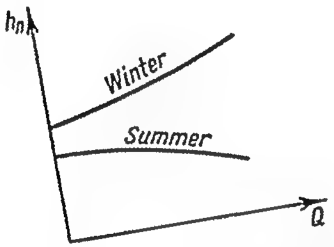

Fig 10-7 The characteristic of the natural ventilating pressure at different seasons

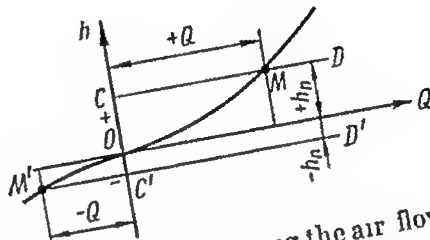

Fig 10-8 Determining the air flow due to the natural draught

of the natural draught is usually drawn as a straight horizontal line.

As with the operation of a fan, the air flow Q provided by the natural draught is equal to the abscissa of the points M or M' (Fig. 10-8) at the intersection of the characteristic of h_n (straight line CD or $C'D$) with the mine characteristic (curve OM or OM').

10-6. THE EFFECT OF THE NATURAL DRAUGHT ON THE RESISTANCE AND THE EQUIVALENT ORIFICE OF THE MINE AND INDIVIDUAL ROADWAYS

This effect was mentioned in Chapter 8. If the natural draught is not included in calculations, but helps the mine ventilation, the mine will be more easily ventilated than the planning would indicate. The smaller the mine head, other things being equal, the greater will be the error.

This feature should be carefully considered when a fan already working at the mine is changed for another one and in choosing the diameter of its impeller in relation to the equivalent orifice of the mine.

CHAPTER 11

FANS

11-1. TYPES OF FAN

Mine fans are installed either at the surface or underground. At the surface they can be either exhaust fans or forcing fans; underground they function as both simultaneously.

The first distinction to make between the radial-flow fans made in the USSR is that some of them have a shock absorber, others

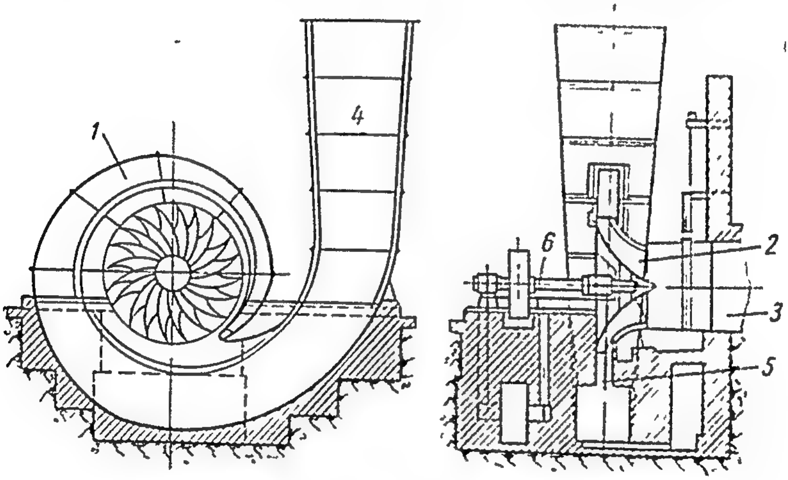

Fig 11-1 Radial-flow fan with shock absorber

do not, secondly, these fans are distinguished by the shape, location and number of their blades

The radial-flow fan with a shock absorber (Fig 11-1) consists mainly of the scroll casing (housing) and the shaft *6*, carrying the impeller *2* with the vanes or blades fixed to it, in these fans the air passing in along the shaft by the intake *3*, enters the impeller, turning through an angle of 90°, simultaneously being twisted in the direction of rotation of the impeller, the air then passes from the vanes of the impeller to the shock absorber *5*, and to the scroll casing *1*, the functions of the shock absorber and casing are to reduce gradually the high flow rate with which the air leaves the impeller,

and simultaneously to reduce the shock losses which are proportional to the flow rate. From the scroll casing the air passes into a gradually widening funnel or diffuser (evase)* 4 (see p. 328) and from it to the atmosphere. In a fan without a shock absorber the air passes directly into the scroll casing from the impeller.

Radial-flow fans are built with the air entering on one side (single-inlet) or on both sides (double-inlet). Some radial-flow types,

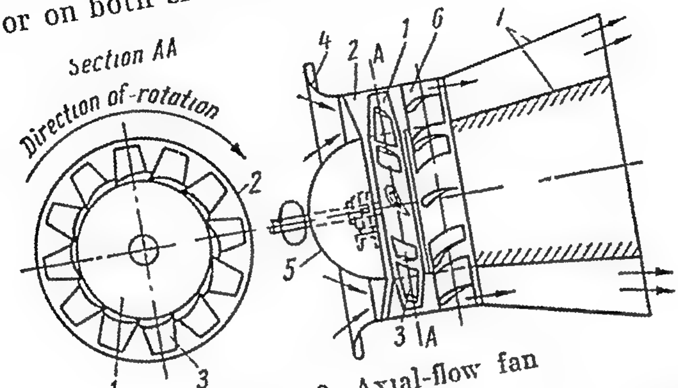

Fig 11-2 Axial-flow fan

the ИГМ single-inlet and the ВЦД double-inlet fans, have guide vanes at the inlet to smooth the air supply

In axial-flow fans (Fig. 11-2) the air passes from the inlet cone 4, along the fan centre line, through the guide vanes into the impeller *1* with blades *3*; from the impeller the air enters the straightening vanes 6, the casing *2*, and the ring-shaped diffuser *7*, continuing to move parallel to the fan centre line. The streamlined nose-piece *5* reduces the pressure losses as the air enters the fan. The guides and straightening devices consist of stationary vanes; their function is to "twist" and then to "untwist" the air stream as it passes out of the rotor, thus increasing the efficiency of the fan

Radial-flow fans are generally distinguished, as explained above, by the location of the vanes; in some of them the vanes are radial, along the rim of the impeller (e g. the ВЦ fans in Fig. 11-3, also the ВЦО), in others they are curved backwards (the ИГМ fan, Fig. 11-4), in a third type they are curved forward (fan with a drum-shaped impeller. Fig. 11-5, often called the Sirocco). The characteristic of a fan with backwardly curved blades (Section 11-3) is described as smooth, i.e. the fan head continually diminishes with increasing air flow. These fans have the highest efficiency. Forwardly curved blades give higher pressures or, which amounts to the same,

* A gradual expansion conversion duct, called an *evase* on exhaust fans and a *diffuser* on blower fans, to convert velocity head to static head and to recover a considerable portion of it

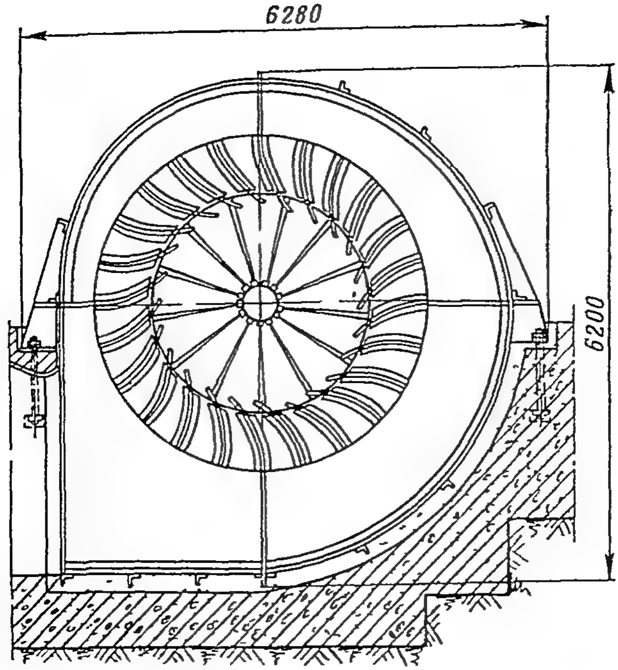

Fig 11-3 Radial-flow fan with radial positioning of the blades

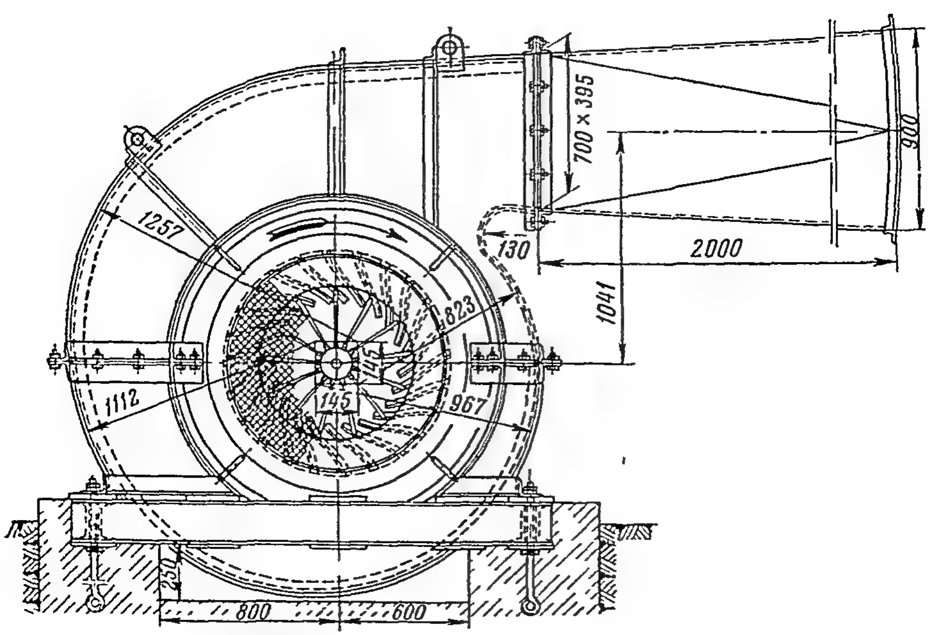

Fig 11-4 Radial-flow fan with blades turned backwards

they enable the required head to be developed at a lower rim speed and with a smaller fan.

Axial-flow fans are distinguished by the shape and number of their blades; thus, in the ЦАГИ fans, series B and BУ, there are 16 blades which are not twisted, and have a tapered side surface; in the ЦАГИ type K-0 6 there are 12 twisted blades, the untwisted blades are hollow, the twisted blades are cast from a light magnesium alloy. The angle of the blades can be changed to vary the flow; the angle of attack of the blades is the angle between the chord of the aerofoil section and the plane of rotation of the impeller (Fig. 11-6). Axial-flow fans are built in one or two stages (with two impellers to increase the head), downstream of each impeller are the stationary guide vanes for straightening the air flow (Fig. 11-7), but the ЦАГИ K-0.6 fan has in addition rotating guide vanes upstream of the first impeller, enabling fine adjustment of the fan without stopping it.

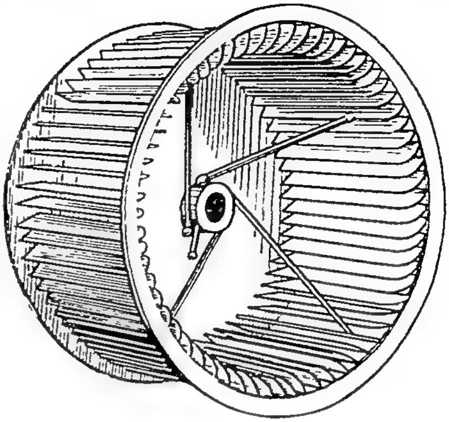

Fig 11-5 Radial-flow fan with blades turned forwards (also known as the drum rotor or Sirocco type)

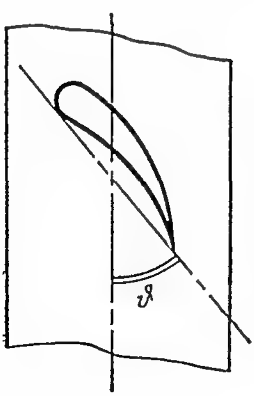

Fig 11-6 Angle of attack of the blades of an axial-flow fan

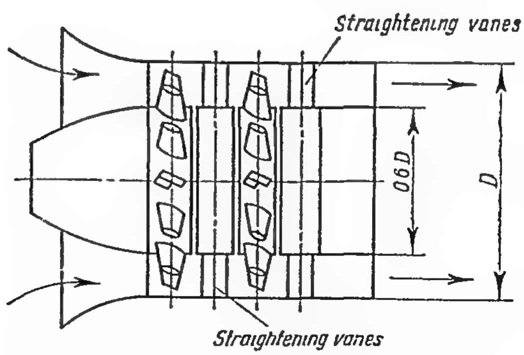

Fig 11-7 Two-stage axial-flow fan

In some countries a new type of high-efficiency mine fan has come into use Thus, in one Ruhr coal mine not long ago a radial-flow fan was introduced, developed by the engineer B. Eck, which has an overall efficiency of above 90 per cent; the impeller of this fan

has a small number of twisted blades, a large relative inlet diameter and in type resembles the drum type of fan; the characteristic of the fan is gently falling.

One axial-flow fan which should be mentioned provides a sharp acceleration to the flow by a restriction of the outer housing or by a taper bushing in it, the fan efficiency being 88 to 89 per cent

The fan performance is characterized by the diameter D of the impeller, its rotational speed (rpm), or the peripheral speed u at the impeller rim, these three values are linked by the equation

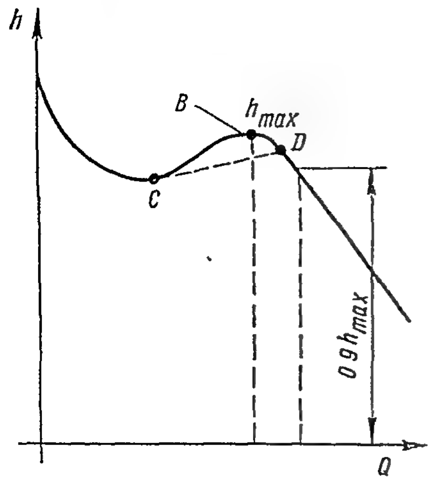

Fig 11-8 Characteristic of an axial-flow fan h_{max} = maximum head created by the fan

$$u = \frac{\pi D n}{60} \text{ m/sec} \qquad (11\text{-}1)$$

Fans are classified according to their function as (a) *main fans* which supply air to the whole mine or part of it, (b) *booster or district fans* which mainly increase the air flow in particular splits, and take the whole air flow to the district; (c) *auxiliary fans* which ventilate dead ends mainly by the use of ventilating ducting and other devices, and pass only a part of a given air stream.

Comparison of radial-flow and axial-flow fans The main differences in operation of these two types of fan are as follows (a) the efficiency (Section 11-2) of the recent Soviet axial-flow fans (e g the ЦАГИ K-0 6) is somewhat higher than that of the new radial-flow fan (ИГМ and ВЦО), i e 0 8 instead of 0 74 to 0 76, in other countries radial-flow fans have been built with an efficiency of 0 92, (b) axial-flow fans until recently had the advantage over radial-flow fans that when their angle of attack was changed, their air flow could be varied within wide limits With the introduction of the hydraulic coupling (Chapter 13, Section 1) as well as the use in radial-flow fans of guide vanes for adjustment, the advantage of changing the angle of attack has been lost, it is true that the guide vanes upstream of the axial-flow fan ensure high performance, but the adjustment is comparatively limited, (c) both types are equally readily reversible, because both require by-pass drifts (Chapter 16); (d) the space occupied by an axial-flow fan of the same capacity as a radial-flow fan is larger because of the horizontal layout of the diffuser, (e) axial-flow fans with a high head (250 to 300 mm) are very noisy,

to reduce the noise a rather inefficient, cumbersome silencer is used, absorbing part of the fan head; (f) radial-flow fans work extremely tranquilly, requiring no repairs for years, which cannot be said of the axial-flow fan since its bearings require replacement every few months; (g) the characteristic of the axial-flow fan is saddle-shaped (with a depression as in Fig. 11-8); the operation of the fan during the transition from the right-hand branch of the characteristic through the summit (or stalling point) to the left-hand branch is unstable, which necessitates the limitation of the working pressure to 0 9 of the maximum pressure at the summit (a proposal of the Soviet Academician V.S. Pak), consequently restricting the range of operation of these fans, (h) for the same reason, parallel operation of axial-flow fans is somewhat less reliable than that of radial-flow fans (Chapter 12, Section 3), (j) for the same reason, the ratio of extremal equivalent orifices with which the axial-flow fan can work efficiently enough is smaller than for the radial-flow fan

11-2. THE PRESSURE AND THE POWER DEVELOPED BY A FAN. FAN EFFICIENCY

Rotating at a certain speed, and working against a given resistance, every fan develops a definite pressure. This total pressure of the fan is spent first of all on overcoming the resistance of the mine airways, secondly on overcoming the resistance of the fan installation (without the impeller) and thirdly on creating the velocity head at the outlet of the fan to the atmosphere

$$h_t = h_m + h_f + h_v \qquad (11\text{-}2)$$

By the term mine we will understand here all the mine workings up to the fan drift, by fan installation, the fan drift, the fan itself and the diffuser (a short, gradually widening funnel through which the air escapes to the atmosphere); its function is described in Section 11-8

In general, h_t will consist of the pressure losses on the exhaust and forcing sides

At the same peripheral speed, the same maximum pressure is developed by impellers of different diameters but of the same type.

The *static pressure* is an important conception, it is measured, for example, in the fan drift of an exhaust fan upstream of its impeller (on the mine side); as was shown in Chapter 3, this pressure is equal to the pressure h_{suct} measured by Pitot static tubes, after the velocity rarefaction has been deducted (Equation 7-16) This is the pressure which is available for ventilating the mine

The *efficiency of a fan*, η, is the ratio of the useful power of the fan $\frac{Qh}{102}$ to the power on the fan shaft N

$$\eta=\frac{Qh}{102N}, \text{ from which } N=\frac{Qh}{102\eta} \qquad (11\text{-}3)$$

Two types of efficiency must be distinguished, the *total* (or *overall*) *efficiency* and the *static efficiency*; the former is always slightly larger than the latter The fan efficiency changes with change of its operating conditions, that is, with change of the discharge at a given speed (rpm) of the impeller It is desirable that the fan efficiency should be at least 0 6.

When the fan works at various conditions and efficiencies, the operating economy is estimated by the weighted mean value of the efficiency, equal to

$$\eta_w=\Sigma Qh/\Sigma Q\frac{h}{\eta} \qquad (11\text{-}4)$$

in which Q and h = the air flow and the head of the fan at different operating conditions

η = the corresponding efficiency at these conditions

The drive used for the fan is invariably an induction motor, although since fans usually work continuously, it would be desirable to install synchronous motors at any power above 100 kW so as to increase the power factor.

The electric motor is connected to the fan shaft (a) directly, to all sizes of axial-flow or radial-flow fans; (b) by a belt drive, the commonest method with radial-flow fans, (c) by toothed gearing.

11-3. THE FAN CHARACTERISTIC

The characteristic of a fan is a curve which expresses the relation between the fan head and the air flowing through the fan at a constant rotational speed (rpm)

At the present level of development of the theory of turbo-machines, fan characteristics cannot be calculated and a fan characteristic must be found by testing it Each fan type has a specific characteristic, *given complete geometric similarity, fans of one type have similar characteristics*

The radial-flow fan characteristic depends on the shape of the blades, and on whether they are forwardly or backwardly curved.

The mine fans made in the USSR have the following characteristics (a) smooth or humpless characteristics for axial-flow fans with a small angle of attack of the blades; (b) humped characteristics for most radial-flow fans, (c) saddle-shaped characteristics

for axial-flow fans having blades with an angle of attack of more than 15°.

Axial-flow fans with saddle-shaped characteristics have the following specific feature. when the equivalent orifice A of the mine diminishes, and the operating conditions move to the summit B of the characteristic there comes a moment (the stalling point B, Fig. 11-8) when with further diminution of A the operating conditions of the fan suddenly drop along the characteristic and the fan continues to work in a stable condition in the region of the trough (point C) with a much smaller output of air.

In the reverse process, i e with increase of A, the operating conditions of the fan move along the dotted line CD until it intersects the right-hand branch of the characteristic. This feature is also found with some fans working in parallel (Chapter 12).

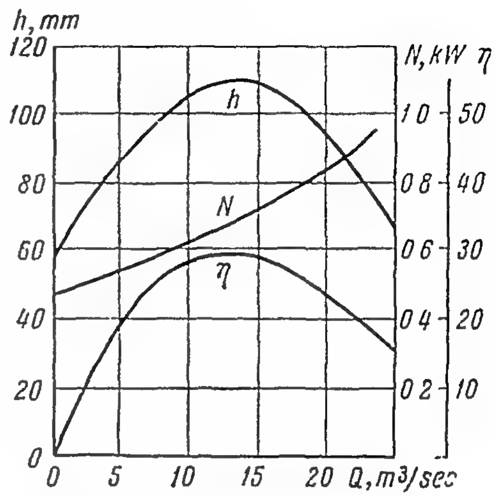

Fig 11-9 Characteristic of a radial-flow fan

Usually in addition to the pressure-quantity curve, on the same diagram (Fig 11-9) efficiency and shaft power are plotted in relation to the air quantity flowing. The power characteristic of all radial-flow fans is rising; with axial-flow fans it is wave-shaped, and usually the power diminishes at large air flows.

11-4. EFFECT ON THE FAN CHARACTERISTIC OF CHANGES IN ITS DIAMETER AND ROTATIONAL SPEED, AND IN THE AIR DENSITY

With change in the number of revolutions n of the fan in unit time, of the diameter D of its impeller and the density γ of the air, changes occur in the quantity of the air flow, the head h. and the power demand N; the fan efficiency also changes, since the point of intersection with the mine characteristic moves along the fan characteristic

If the values Q_1, h_1, and N_1 are known for a rotational speed n_1, a diameter D_1, and a density γ_1, then for converting the characteristic at a different fan speed n_2, diameter D_2, or density γ_2. the formulas in Table 11-1 are used.

With these formulas it must be remembered that they apply only to fully geometrically similar fans of the same type.

TABLE 11-1 Equations for Converting Fan Characteristics at Varying Values of Diameter, Specific Weight of Air, etc.

Conversion for γ	Conversion for n	Conversion for D	Conversion for γ, n and D
$Q_2 = Q_1$	$Q_2 = Q_1 \frac{n_2}{n_1}$	$Q_2 = Q_1 \left(\frac{D_2}{D_1}\right)^3$	$Q_2 = Q_1 \frac{n_2}{n_1} \left(\frac{D_2}{D_1}\right)^3$
$h_2 = h_1 \frac{\gamma_2}{\gamma_1}$	$h_2 = h_1 \left(\frac{n_2}{n_1}\right)^2$	$h_2 = h_1 \left(\frac{D_2}{D_1}\right)^2$	$h_2 = h_1 \frac{\gamma_2}{\gamma_1} \left(\frac{n_2}{n_1}\right)^2 \times \left(\frac{D_2}{D_1}\right)^2$
$N_2 = N_1 \frac{\gamma_2}{\gamma_1}$	$N_2 = N_1 \left(\frac{n_2}{n_1}\right)^3$	$N_2 = N_1 \left(\frac{D_2}{D_1}\right)^5$	$N_2 = N_1 \frac{\gamma_2}{\gamma_1} \left(\frac{n_2}{n_1}\right)^3 \times \left(\frac{D_2}{D_1}\right)^5$
$\eta_2 = \eta_1$	$\eta_2 = \eta_1$	$\eta_2 = \eta_1$	$\eta_2 = \eta_1$

Note In the conversion for diameter the peripheral speed u varies as D_2 D_1, since the rotational speed (rpm) n is constant

Using these formulas (Table 11-1) it is possible from characteristics obtained experimentally for one fan speed (rpm) or for one diameter of fan impeller, to obtain characteristics at any fan speed for any fan For example, a characteristic is obtained at a fan speed of $n = 400$ rpm, and it is required to re-plot it for $n = 500$ rpm, we take an arbitrary point from this given characteristic and note its co-ordinates, e g $h = 80$ mm, $Q = 60$ m³/sec, in accordance with the conversion formula for n, the air flow and the head at $n = 500$ rpm will be

$$Q_2 = 60\left(\frac{500}{400}\right) = 75 \text{ m}^3/\text{sec}$$

$$h_2 = 80\left(\frac{500}{400}\right)^2 = 125 \text{ mm } H_2O$$

Repeating these calculations for other arbitrary points of the characteristic, we obtain the co-ordinates of several points at $n = 500$ rpm, and by joining them we obtain the required characteristic.

The first column in Table 11-1 shows that in designing a fan for work at a high altitude above sea level, the fan must be selected for delivering the air flow needed in normal conditions of operation,

but with a pressure modified as follows

$$h = h_0 \frac{\gamma}{1.2}$$

in which γ = the specific weight of the air at the point of installation of the fan

h_0 = the pressure calculated for normal conditions of operation

Fig. 11-10 shows, by way of example, how the characteristics of a radial-flow fan will change (a) with changing peripheral speed, at a constant diameter D of 1.2 m, and (b) with changes of D at a constant peripheral speed u = = 30 m/sec; with changing peripheral speed (rpm), the characteristics move upwards and to the right; with changing diameter of impeller and the same peripheral speed, they move to the right; the maximum pressure remains unchanged.

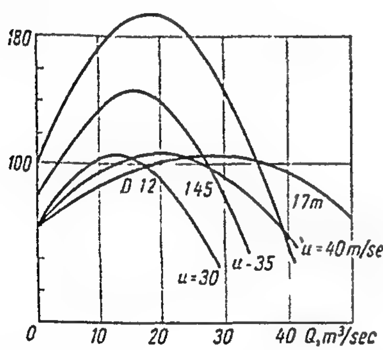

Fig. 11-10. Change in the characteristic of a radial-flow fan with varying rim speed and rotor diameter, D

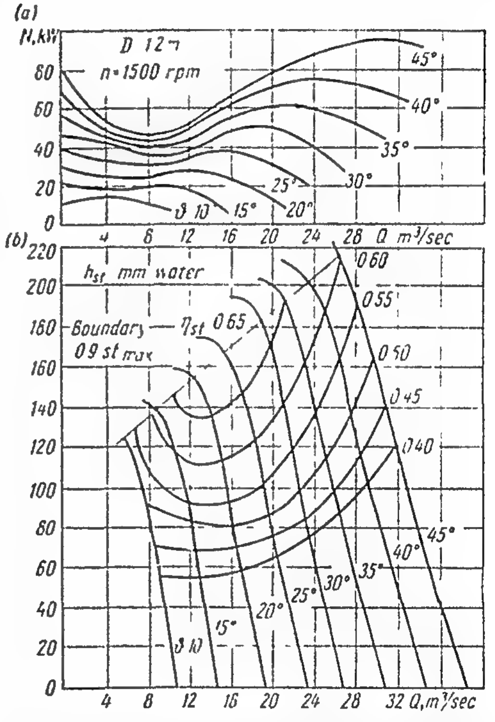

Fig. 11-11. Variation in the characteristic of an axial-flow fan with change in the angle of attack of the blades

The characteristic of an axial-flow fan changes with changing angle of attack of the blades, moving upwards and to the right with increasing angle of attack (Fig. 11-11*a* and *b* referring to an axial-flow single-stage fan, series K, with a long diffuser and D = 1.2 m).

Apart from these characteristics, curves are sometimes drawn which show the change of air flow through the fan with changing equivalent orifice of the mine; these are readily obtainable from the usual characteristics if curves are plotted for various values of equivalent orifice, and the necessary air flows for them are found

11-5. THE OPERATION OF MINE FANS

The function of the mine fan is to send into the mine enough air to maintain its standards of air purity. Not every fan, however, can meet this condition; the dimensions of the fan and its rotational speed (rpm) must be selected so that *it can give the necessary air quantity against the mine resistance and that it does this work economically*. The problem consequently can be stated as follows: knowing the resistance R or the equivalent orifice A of the mine, select a fan satisfying both these requirements.

A simplified method of choosing a fan is given on p. 534. More accurate methods are described in courses on mine fans.

The operating condition of the fan, as explained in Chapter 8 (p 271), is defined by the point of intersection of the mine characteristic with that of the fan, the air quantity provided by the fan at a known rotational speed (rpm) is completely defined by the resistance which the fan must overcome. A large fan installed at a neglected mine with long and narrow airways, that is, having a very high ventilating resistance, may provide less air than a smaller fan working at a mine with short wide airways. Occasionally in the literature and in ventilation projects we find the expression "a fan of 30 m³/sec output", or "a 5,000 m³/min fan", without indication of the pressures corresponding to these outputs; such statements make no sense because such fans can provide either 20 or 40 m³/sec, or 3,000 or 8,000 m³/min, respectively, depending on the resistance and the fan speed (rpm). We must understand quite clearly that although the characteristic of the mine has nothing to do with the fan and the characteristic of the fan does not depend on the particular mine at which it is installed, *the question of the quantity of air delivered by a fan can be solved only in connection with the resistance which the fan must overcome* Any change in the mine resistance resulting, for example, from the installation of a regulator, or a change in the ventilation layout, from the introduction of a new production face, or the widening of the workings, or a rock fall, etc , brings about changes in the ventilation conditions, i e. in the values of h and Q

The main fan is usually installed at a shaft which is closed at the top by an airtight building (or, in the case of a small pit, simply by trapdoors) Both through the airtight building and through the trapdoors there will be inevitable air leakages; according to current standards, the amount of leakage should not exceed 10 per cent of the quantity entering the mine If we denote the percentage of leakage by p, the fan should be chosen so (a) that the fan delivery $Q_f = 1.1\ Q_m$; (b) that the total resistance against which the fan

works should be equal to

$$R_t = kR_m + R_f \tag{11-4}$$

in which R_f = the resistance of the fan installation
k = the leakage factor (see p. 282).

It follows from Equation (11-4) that R_t can be larger or smaller than R_m; if the leakage predominates, then R_t will be less than R_m, if the resistance R_f is relatively large, then R_t will be larger than R_m.

11-6. THE FAN INSTALLATION

As pointed out, the mine fan installation ordinarily consists of the fan itself, the fan drift, the diffuser, and the fan drive.

The main requirement for the fan installation is to minimize its pressure losses, including the loss of velocity head at the outlet

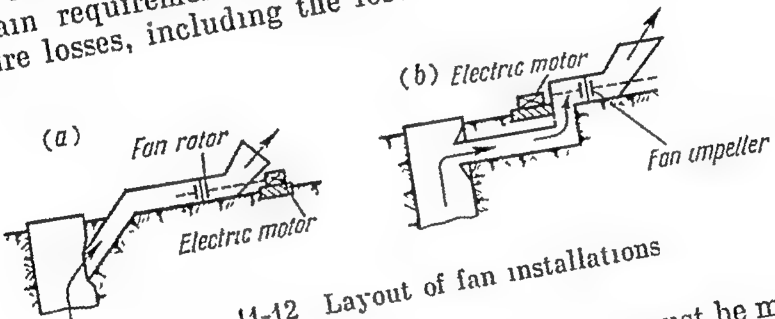

Fig 11-12 Layout of fan installations

to the atmosphere; in other words the value of R_f must be minimized

Fan installations often do not satisfy these requirements, in particular: (a) the fan drifts are narrow, have unnecessary bends, are cluttered up with rubbish, and join the shaft at a right angle; (b) sometimes the fan drift has a bend directly upstream of the rotor of an axial-flow fan, consequently the air stream is pressed to one side of the drift, and the air falls practically onto one side of the impeller; in such conditions the impeller operation is far from normal (Fig. 11-12a, b).

The resistance of the fan installation is hereinafter understood to mean its total resistance: that of the bend from the shaft into the fan drift, that of the fan drift itself, and of its bends up to the fan impeller, that of the bearing supports of an axial-flow fan, of the diffuser and of the velocity head at the outlet into the atmosphere (see Section 11-8).

Equation (11-5) below gives, with sufficient accuracy for practical purposes, the resistance R_f of a fan installation with a fan of diameter D.

$$R_f = a\frac{\pi}{D^4} \tag{11-5}$$

in which the factor a is.

axial-flow fans with a smooth fan drift	0 05
axial-flow fans with bent fan drift and without guide vanes	0 10
double-inlet radial-flow fans without shock absorber and with a bend to the diffuser	0 075
double-inlet radial-flow fans without shock absorber and with a pyramidal diffuser	0 055
fans with shock absorber, and drum type fans (Sirocco), approx	0 040

Methods of Reducing the Resistance of the Fan Installation

1 The resistance of the fan drift can be reduced by (a) cleaning the fan drift if it is cluttered up, and removing any props, beams, etc , (b) smoothing the corners, if the fan drift has bends; particularly harmful are sharp corners near the inlets to a double-inlet fan without shock absorber, (c) installation of guide vanes at bends (see Chapter 8), (d) building a smooth junction between the shaft and the fan drift, (e) re-lining and widening the fan drift.

2 Installation of a diffuser of suitable shape and dimensions (see Section 8).

11-7. ANALYSIS OF THE FACTORS WHICH AFFECT THE OPERATION OF MINE FANS

These factors include.

1 *The diameter of the fan impeller and its peripheral speed. Generally, the larger the impeller diameter and peripheral speed or rotational speed (rpm) the larger will be the air quantity put through the fan*, however, at small equivalent orifices A, i e with a steep characteristic, changing one fan for another of larger diameter will give little or no gain in air quantity. The value of the peripheral speed u is limited by strength considerations if the peripheral speed is too high the impeller will break up, or the blades will fly off it, in addition it must be remembered that the higher the peripheral speed of an axial-flow fan, the more noise it will make

2 *The total equivalent orifice A against which the fan works* The larger the orifice, the more air will the fan give, the less the slope in the working sector of the characteristic, the greater the increase of the output, but the steeper is this sector, the less will be the relative decrease of fan output with decrease of equivalent orifice

3 *The sector on the characteristic which corresponds to the given operating conditions of the fan* The working sector of the characteristic should be on the right-hand branch of the characteristic, slop-

ing downwards; this particularly concerns axial-flow fans in which the fan may stall during the transition of its operating conditions across the hump to the left-hand side of the characteristic. Stalling can easily be recognized by the characteristic "intermittent" operation of the fan; such unstable operation cannot be tolerated because it hinders the operation of the whole ventilating installation Sectors even further to the left, i e. in the valley, have a low efficiency, although they are stable.

4. *Efficiency of the fan*. As Fig. 11-9 shows, the maximum value of the fan efficiency is approximately at the hump of the characteristic, consequently, for economy *the operating sector should be as close to this crest as possible and to the right of it* The smaller the diameter of the fan impeller, the closer to the ordinate axis will the crest of the hump be; on the other hand, the smaller the equivalent orifice, the further to the left will the mine characteristic be; therefore, so that the two characteristics shall intersect close to the hump of the fan characteristic in the region of its most favourable operating conditions, the *impeller diameter should be small if the equivalent orifice of the mine is small*

It is wrong to believe that the more difficult the mine is to ventilate the larger should be the fan installed, because the crest of the fan characteristic will be to the right of the mine characteristic and therefore these curves will intersect either in the area of unstable operation or in the area of low efficiency, in addition the output of a large fan is often smaller than that of a small fan.

5. *The resistance of the fan installation*. It follows directly from Equation 11-4 that the resistance of the fan installation itself has a considerable and sometimes deciding effect (with a large value of R_f) on the value of the total resistance against which the fan works. *The larger the equivalent orifice of the mine and the smaller its resistance, the smaller should be the fan installation resistance* In practice there are fans in which the fan installation resistance R_f is larger than R_m, the mine resistance, there is no need to emphasize how much this reduces the fan output.

6. *Short-circuiting at the top of the upcast shaft* An examination of more than 150 fan installations has shown that in most of them the leakage by far exceeded 10 per cent; in some of them more air was short-circuited than went into the mine The short-circuits reduce the resistance against which the fan works, and the fan gives more air, but less of it goes through the mine Leakages also result in a large unproductive consumption of energy.

The smaller the equivalent orifice of the mine for the same resistance of the shaft collar, the greater will be the percentage of leakage.

11-8. DIFFUSERS AND EVASES

The air velocity at the discharge of the fan into the atmosphere is generally large, sometimes exceeding 30 m/sec. The velocity head h_v created by this air is equal to $\frac{v^2}{2g}\gamma$ mm H_2O, and the energy carried by the air is equal to $Q\frac{v^2}{2g}\gamma$ kg-m/sec, in which Q is the fan output, and v is the average velocity.

This power is obviously completely lost to the main work of ventilating the mine It is easy to see that if a windmill was set in front of the fan outlet, it could work on the power provided by the fan.

The higher the air flow rate, the greater will this loss be, it can reach 35 to 40 per cent or more of the power consumed by the fan

It is therefore clear that *to reduce useless power losses and increase the useful amount of air delivered by the fan, the air flow rate at the outlet to atmosphere should be reduced* This is achieved by a comparatively short widened tube, connected to the fan casing, and called the *diffuser* (*evase*)*

Diffusers are most often pyramidal in shape, rarely conical.

The pressure head loss $\frac{v^2}{2g}\gamma$ at the outlet from the fan casing to the atmosphere can be taken as the pressure lost by air passing through a local resistance R'_{out} with a coefficient $\xi = 1$**; from Equation (7-20) it equals $\frac{0\ 0612 \times 1\ 0}{S^2_{out}}$ in which S_{out} is the area of the outlet from the fan casing, similarly, using S_{dif} for the outlet area of the diffuser we obtain

$$R''_{out} = 0\ 0612\,\frac{1\ 0}{S^2_{dif}}$$

Denoting the resistance of the diffuser by R_{dif}, the pressure loss in the diffuser will be caused by (a) overcoming its resistance, and (b) transforming the dynamic pressure into static pressure, in diffusers with a right-angled bend there is also a pressure loss at the bend

The purpose of installing a diffuser consists in reducing the air velocity at the outlet to the atmosphere so that the resistance R'_{out} at the outlet from the fan casing at high air velocity shall be reduced

---

* See footnote to 11-1 (p 315)

** In reality, because of the non-uniform distribution of the air velocity across the diffuser outlet, the pressure loss is equal to kh_v in which λ is the kinetic energy coefficient, the value of this coefficient can reach 2 and even more in diffusers with a right-angled bend or with air breaking away from the sides, but in this instance ξ is larger than 1

to the lower total value of $R_{dif} + R''_{out}$ leaving the diffuser at low air velocity.

The installation of a diffuser is advisable subject to the condition that

$$(R_{dif} + R''_{out}) < R'_{out} \tag{11-6}$$

Reduction of the air velocity at the outlet from the diffuser to the atmosphere can be achieved by the following two methods or a combination of them: (1) increasing the length L of the diffuser while maintaining the same angle of flare in it, and (2) increasing the angle of flare while maintaining the same length L of the diffuser.

Usually the angle of flare is 8 to 10°, and $S_{out} \cdot S_{cas} = 3\text{-}4$, in which S_{out} and S_{cas} are the areas of the outlets from the diffuser and from the fan casing.

The horizontal diffuser found in some fans should be placed to face the prevailing wind; a baffle should be placed 2 to 3 m in front of it so as to prevent the wind blowing into the diffuser during storms when not only could the air flow be greatly reduced, but it might even be reversed.

CHAPTER 12

USE OF SEVERAL FANS SIMULTANEOUSLY. VENTILATION BY NATURAL DRAUGHT AND THE FAN

Ventilation by one fan is in practice comparatively rare; much more often several fans work together at a mine, the main fan and the boosters as well as the natural draught.

Simultaneous operation at one mine of several fans, or of a fan and natural draught, or of several simultaneous but different natural draughts, each differ from the operating conditions with a single fan or natural draught alone; this chapter will deal with the commonest methods of ventilating mines by mechanical fans and natural draught

12-1. COMPOSITE CHARACTERISTICS OF FANS

In the operation of fans it has been accepted until now that the air first moves through the fan from the inlet towards the diffuser, and, secondly, that the air pressure in this direction increases, i e there is a suction at the inlet, and there is a pressure at the diffuser which is higher than that at the inlet

Figure 12-1 is a graphical representation of the work of the fan, with axes h and Q, where the pressure difference $p_2 - p_1$ between the diffuser and the inlet, and the air flow from inlet to diffuser are taken as *positive* and measured upwards from the abscissa axis and to the right from the ordinate axis

We will consider these conditions of operation as "*normal*".

How should we consider points which are below the abscissa axis and to the left of the ordinate axis, that is, in areas with negative pressures and air flows?

Evidently such operating conditions would be the reverse of "normal", i e when the direction of air flow, for the same direction of rotation of the impeller, is from the diffuser to the inlet, and the pressure at the inlet exceeds the pressure at the diffuser When a fan is working alone and without a natural draught, this situation is evidently impossible

Let us, however, imagine that two fans are working together in such a way that the diffuser of one of them is connected to the inlet of the other (Fig 12-2*a*), these fans are said to be in series, in this instance, under the pressure created by the upstream fan, the

pressure p_1 at the inlet to the downstream fan may be higher than the pressure p_2 in the diffuser, and consequently with positive flows, the pressure difference $p_2 - p_1$ at the downstream fan will become negative, that is, it will correspond to the conditions of the points a, a, (Fig. 12-1) below the axis of abscissas in the fourth quadrant

Let us now take two fans and connect their inlets to the same circuit (Fig 12-2b), these fans are said to be in parallel In this instance,

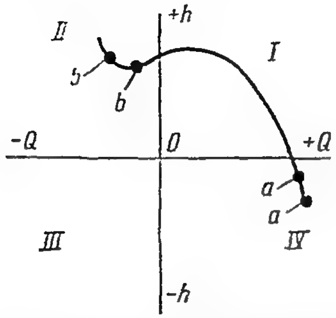

Fig 12-1. Analysis of working conditions of a fan

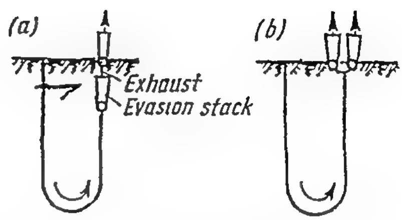

Fig 12-2 Fans working (a) in series, (b) in parallel

with a large difference between the pressures h_1 and h_2 of the two fans, it may happen that the fan developing the larger pressure may "pull over" the fan with the smaller pressure and the air will be sucked out of the atmosphere into the diffuser of the smaller fan, passing into the inlet of the larger fan; that is, with a positive pressure $p_1 - p_2$ at the first fan, its output will become negative, implying that it will correspond to the conditions of points c, c (Fig 12-1) to the left of the ordinate axis in the second quadrant Thus, together with "normal" operation in the first quadrant, operation in the second and fourth quadrants is also possible.

Section 12-6 will show how operation is also possible in the third quadrant

12-2. FANS OPERATING IN SERIES

Fans are found in series in the following layouts:

1. Simultaneous operation of fans at one shaft (Fig 12-3a).
2. Simultaneous operation with one fan at the surface and the other underground (Fig 12-3b)
3. Simultaneous operation of fans placed at different shafts with one working as a blower and the other as an exhaust fan (Fig. 12-3c).
4. Simultaneous operation of two or more fans on a single line of ventilation ducting with partial ventilation (Fig. 12-3d).

With two fans in series, the quantity of air passing through each of them is evidently the same, i e. $Q_1 = Q_2$, and the total pressure

will be equal to the sum of the heads developed by the two fans, i e $h_t = h_1 + h_2$

Thus, to obtain the composite characteristic of two fans connected in series, it is necessary to add the heads of the individual characteristics for the same quantity of air.

The two characteristics of the fans are therefore drawn on one diagram (Fig 12-4) and a number of arbitrary vertical lines, *1-1*, *2-2*, etc , are drawn, intersecting these characteristics; then, with dividers, the intercepts between the abscissa axis and each of these curves are added together in the following way intercepts ab and ac, $a'b$ and $a'c'$, etc ; then, joining the points d, d', d'', etc., of the summed intercepts, we obtain the required composite characteristic.

The total air flow from these two fans together is the abscissa

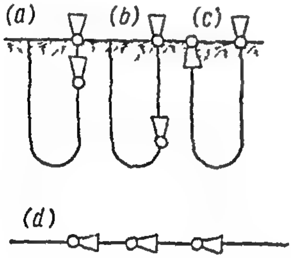

Fig. 12-3 Fans in series
(a) at the surface at the same shaft, (b) one fan at the surface, the other underground (c) fans in series at different shafts, (d) along a line of ventilation ducting in auxiliary ventilation

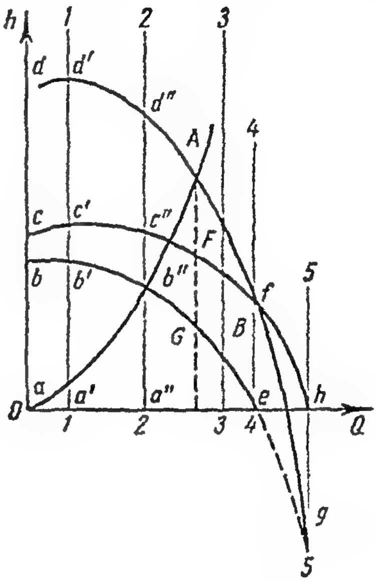

Fig 12-4 Drawing up an overall characteristic for two fans in series

of the point A of intersection of the composite characteristic of the two fans with the characteristic of the mine or air circuit. The ordinate of this point will be the total pressure If the two fans are similar and rotate at the same speed, operation in series always gives more air, the increase depending on the shape of the fan characteristic, if the characteristic is smooth (without humps or valleys) then with increasing mine resistance the effectiveness of the joint operation of the fans increases to a maximum of about 40 per cent; if the fan characteristics are humped (radial-flow fans) or saddle-shaped (axial-flow fans), the maximum increase of air flow will be about 45 per cent, reaching 80 per cent when the mine character-

istic intersects the composite characteristic at its highest point. With further increase of the mine resistance, i e with a shift of its characteristic to the left, the effectiveness of joint operation diminishes and drops to 40 per cent as for fans having a smooth characteristic *The total discharge of jointly operating fans is always less than the sum of their discharges when operating individually*, i e. Q_t is less than $Q_1 + Q_2$

If the characteristics of the two fans are different, operation in series will always be profitable with a large mine resistance R_m and

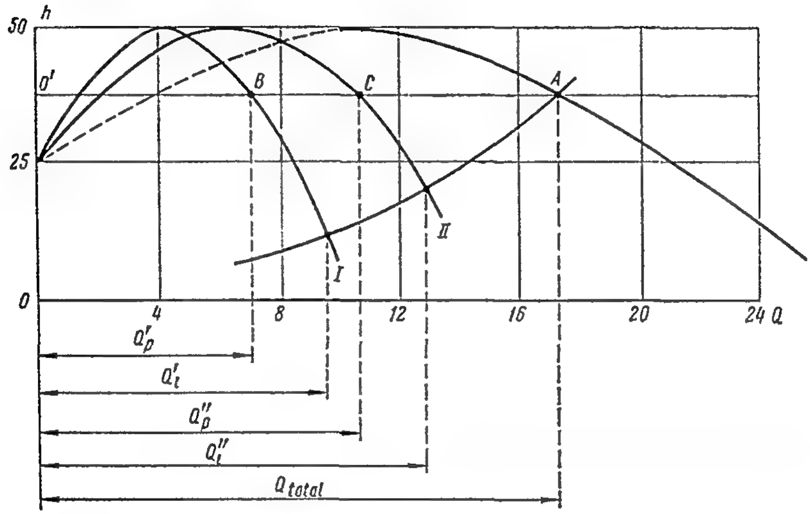

Fig. 12-5 Drawing up an overall characteristic for two fans in parallel

may be unprofitable with a small value of R_m when the mine characteristic intersects that of the large fan to the right of the limiting point B where the same characteristic cuts the composite characteristic of the two fans (see Fig. 12-4).

As the graph shows, the discharge of the large fan at this point is equal to the discharge of the small fan, which is developing a smaller pressure and not working against the mine resistance, and without velocity pressure at the outlet, in conditions to the left of the point B, the work of the small fan will be expedient, in conditions to the right it will be harmful, and in the conditions of point B it will be useless

The partial heads h'_p and h''_p developed by the two fans are the ordinates of the points F and G of intersection of the total head h_t with the individual characteristics of each fan (Fig 12-5).

In conclusion we should point out the following. because the resistance of the mine changes continuously, the working of fans

in series during the course of mining can sometimes become unprofitable (with increase of the equivalent orifice of the mine) and, conversely, profitable with increase of the overall resistance of the mine

12-3. FANS IN PARALLEL AT ONE SHAFT

In the parallel operation of fans, i e. with fans connected directly to the same point of an airway, the pressures will be the same (neglecting the difference in resistances in the fan drifts and diffusers) but the air quantities may be different

$$h_1 = h_2; \quad Q_1 \neq Q_2, \quad Q_t = Q_1 + Q_2$$

To obtain the composite characteristic of two fans working in parallel it is necessary to add the air flows of the individual characteristics at the same pressure, i e choosing arbitrary pressures the abscissas from the characteristics at the points of these pressures are added On the diagram with the two characteristics drawn (Fig 12-6) a number of arbitrary horizontal lines are drawn, and the intercepts between the ordinate axis and each of the characteristics are added together (e g $O'B$ and $O'C = O'A$), the points being joined to form the composite characteristic

The total air flow Q_t which will be given by the two fans is the abscissa of the point A of intersection of the composite characteristic of the two fans with the characteristic of the mine

The discharge from each fan during joint operation, or, stated differently, their partial discharges Q'_p and Q''_p will be equal to the abscissas of the points B and C of intersection of the individual characteristics of each fan with a horizontal straight line through the point A

From the drawing it can be seen that the partial discharges are smaller than the individual discharges, $Q_p < Q_i$, consequently, the total air flow during parallel operation of the fans will be less than the sum of their individual discharges

$$Q'_p + Q''_p = Q_t < Q'_i + Q''_i$$

In the preceding section, considering several fans in series, independently of the shapes of the characteristics of the fans working together, and their diameter and peripheral speed, we always obtained one operating condition, i e a single point where the composite fan characteristic cuts the mine characteristic

In parallel operation, as we shall see below, stable operation of two fans together may occur as with fans in series, but there are instances when the composite characteristic cuts the mine characteristic not at one point, but at two or three points, and hence

several different operating conditions are possible. We will look at these instances separately

When the composite characteristic of the fans is smooth and not humped, operation at any equivalent orifice of the mine will be stable

But *if the fans are similar and their impeller speeds are the same, parallel operation will always be profitable, the more so the larger*

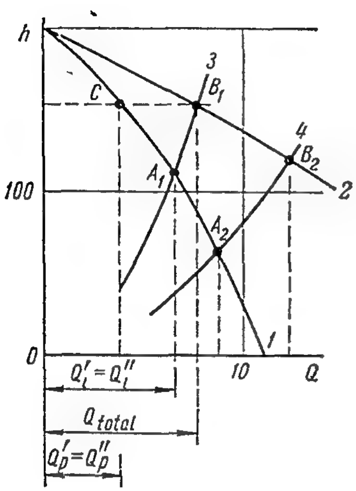

Fig 12-6 Graphs for two similar axial-flow fans in parallel

1—individual characteristic of the two similar fans, 2—overall characteristic of the two fans, 3—characteristic of a mine with a small equivalent orifice, 4—characteristic of a mine with a large equivalent orifice, A_1 and A_2 are points corresponding to the fans working separately, B_1 and B_2, ditto with the fans in parallel, C indicates the separate discharges of the fans in parallel

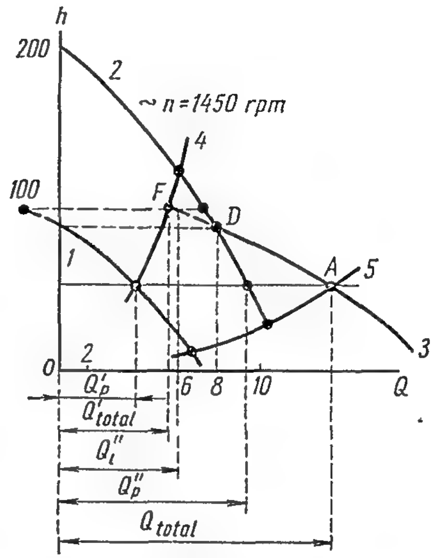

Fig 12-7 Graphs for two different axial-flow fans in parallel

1—characteristic of the small fan, 2—characteristic of the large fan, 3—overall characteristic of the two fans in parallel, 4—characteristic of a mine with a small equivalent orifice, 5—ditto, with a large equivalent orifice, D is the point which is on the boundary between favourable and unfavourable operating conditions of the fans

the equivalent orifice of the mine, the larger the impeller diameter, and the lower the peripheral speed (Fig 12-6)* The partial discharge of each fan is equal to half the total If their rotational speeds are different, whether the fans are similar or of different types, parallel operation of the second fan may be profitable, unprofitable, or harmful, depending on the equivalent orifice of the mine (Fig 12-7, conditions at points *A*, *D*, and *F*).

* Fig 12-6 shows the operation of two axial-flow fans of ЦАГИ series B, with $D = 1.2$ m, $n = 1\,450$ rpm and an angle of attack of 10°

The point on the composite characteristic which separates profitable from harmful operation is the point D where the composite characteristic intersects the characteristic of the large fan (Fig. 12-7)

For humped characteristics, all that has been said on the parallel operation of fans with non-humped characteristics holds good In addition the following also applies the shapes of humps, i.e areas where a single pressure corresponds to several discharges, are repeated in the shape of the composite characteristic, and the

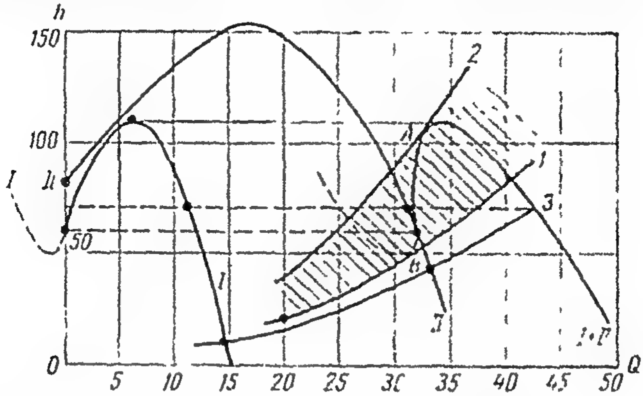

Fig 12-8 Drawing up an overall characteristic for two fans in parallel, with humped characteristics

I—characteristic of the small fan, *II*—characteristic of the large fan, *I + II*—overall characteristic of the two fans, *3*—characteristic of the mine corresponding to steady operating conditions of the two fans in parallel, *1* and *2*—characteristics of the mine corresponding to non-steady working conditions

result or the adding of the abscissas of the individual characteristics is a curve which is not simple, but complex, having several inflections (Fig 12-8). But these complex characteristics must cut the mine characteristic, at several particular values of the equivalent orifice of the mine, not at one, but at two or three points, since each intersection corresponds to a possible condition of operation of the fans, we come to the inevitable conclusion that sometimes with fans in parallel, several different conditions of work are possible

The establishment of particular operating conditions of joint operation of the fans depends on the sequence of starting the separate fans, after the desired conditions are set up, the fans will continue to be stable, although with reduction or increase in the resistance of the mine, i e a downward or upward shift of its characteristic, the mine characteristic may become tangential to the composite characteristic of the two fans (A and B in Fig 12-8); then the operating conditions become unstable, after which with further reduction or increase of the mine resistance, the conditions will again become steady,

independently of the sequence of starting of the fans which are working in parallel

Similar conclusions result from an analysis of the parallel work of fans with saddle-shaped characteristics (axial-flow at angles of attack of about 15°) with the difference that because of the unstable zone of operation even with single-stage axial-flow fans (p. 318) instability is more likely.

We can therefore draw the following conclusion: *in the parallel operation of two fans with similar smooth characteristics there will only be positive discharges, in the parallel operation of two fans with different smooth characteristics, both positive and negative discharges are possible,* in the parallel operation of two fans with humped or saddle-shaped characteristics, stable positive and negative conditions of operation are possible; moreover, for the hatched area on Fig. 12-8 indicating values of the mine resistance in which various operating conditions are possible, there will be instability at the limits of the area.

Provided that the anticipated operating conditions will be stable enough, and the fans will be working above or below the zone of several possible operating conditions, there is no reason to forbid the parallel operation of two fans at one shaft. The increase of discharge may reach 80 per cent.

In contrast to series operation, parallel operation of fans, as shown in Fig. 12-7, will be profitable in mines with large equivalent orifices and unprofitable in mines which are difficult to ventilate.

Joint operation of fans at one shaft can be used for ventilating large deep mines, when the choice of a single fan is made difficult either because of the high pressure drop or because of the large quantity of air required.

12-4. FANS IN PARALLEL AT DIFFERENT SHAFTS

This type of parallel operation of fans is widely used in practice for ventilating mines by fans at the mine boundaries, installed at small pits or drifts

From the instance above, of two fans working in parallel at one shaft, the layout to be considered below differs by the presence of the sections *BC* and *BD* between the fans (Fig. 12-9*a*, *b*).

The first solution of this problem was given by Associate Professor D.F. Borisov of the Leningrad Mining Institute in 1935.

Other methods of solving the problem of the parallel operation of fans were given by Professor B. S. Pak and other authors. Below we give the method of D. F. Borisov which in the authors' opinion is the best method for understanding the parallel operation of fans.

and the composite characteristic $I' + II''$, $EFGH$, of the fans is drawn; the intersection M is found, where the composite characteristic cuts the characteristic OMB of the common section AB (in this instance $R_t = 0.03$ kilomurg); the abscissa of this point is equal to the total air flow; through the point M a horizontal line

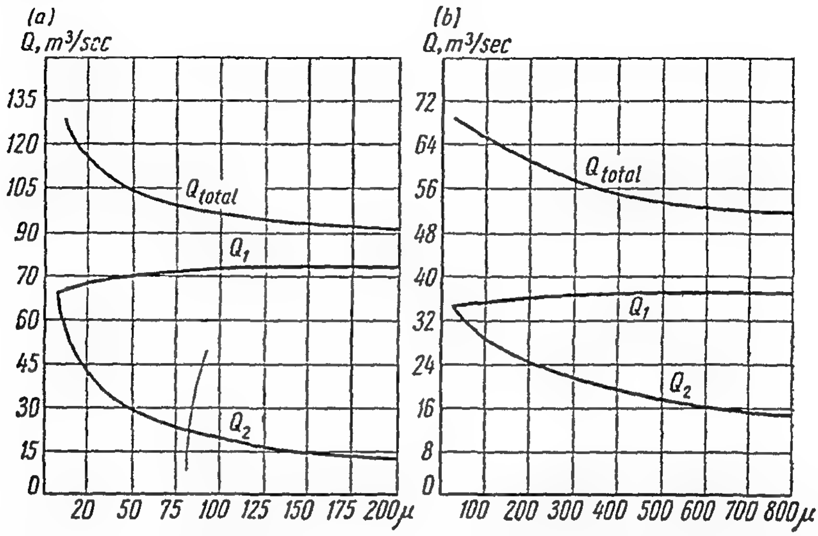

Fig 12-10 Variation of output from fans operating separately and jointly, working in parallel against a varying resistance.
(a) radial-flow, (b) axial-flow

is drawn until it intersects points *1* and *2* on the reduced fan characteristics; the abscissas of the intersections *1* and *2* will be equal to the air flows Q_1 and Q_2 from the fans.

The total head of the fans is obtained by adding the heads of the sections BC and BD to the ordinates of points *1* and *2* by extending the ordinates of these points until they meet the individual characteristics of the fans at points *1* and *2'*. These points indicate the operating conditions of each fan.

The intersection M of the composite reduced characteristic EH with the characteristic of the common section of the ventilation circuit may lie along various lengths of the curve $I' + II'$.

First of all, it can be in a length which corresponds to the total positive lengths of the reduced individual curves I'-I and II''-II'' (length EFG); in this case both fans will give positive discharges (assuming that the discharges will be sufficient for ventilating each wing of the mine).

Secondly, if the impellers differ greatly in diameter or in peripheral speed, it may happen that one fan so much surpasses the

other that the discharge of the smaller fan, although positive, may prove inadequate

Thirdly, the intersection can fall at such a point (in this case at point *G*) of the composite reduced characteristic that it corresponds to the zero conditions of the small fan and in this case the fan will give no air

Finally, a fourth variant is possible, with the intersection in the part *GH* of the composite characteristic which corresponds to negative operation of the small fan, and the large fan will suck air through the small one

A special case occurs when the composite characteristic at a certain length is loop-shaped, as in this instance, and the characteristic of the common section intersects the overall characteristic exactly at this point

As with the operation of two fans at one shaft, several operating conditions of the fans are possible in the area limited by the tangents to the points of inflection of the composite characteristic (Fig 12-9*d*) and at the boundaries of this zone operating conditions are unstable Compared with parallel operation at one shaft, the difference lies in the fact that the mutual effect of one fan on the other is smaller because the whole mine lies between the two fans.

The mutual effect of one fan on the other or, which is the same, the suction of one fan through another, becomes more likely. (a) when the resistance of the common section increases; (b) when the resistance of any one of the sections diminishes; (c) when the rotational speed (rpm) of the fan increases or one fan is changed for a more powerful one.

Action must be taken to reduce the suction when, after one of the fans placed at the boundary of the take has been changed for a more powerful one, the air flow from the second fan becomes inadequate or ceases completely.

The suction occurs because, *at the junction of the splits, the head of the fan which is sucking the air is larger than the head developed at the same point by the other fan* To reduce the suction the head of the first fan should be reduced and that of the second fan increased. This can be achieved

In the first fan· (a) by decreasing the rotational speed (rpm); (b) by changing the angle of attack of the impeller blades in an axial-flow fan, (c) by building a regulator or by lowering a gate in the fan drift; (d) by a slight increase of leakage from the surface.

In the second fan (a) by increasing the rotational speed (rpm), (b) by changing the angle of attack of the impeller blades in an axial-flow fan; (c) by reducing the leakage from the surface; (d) by reducing the resistance of the fan installation; (e) by removing the props

It is possible that the required air distribution cannot be achieved by any of these measures; when this occurs, the second fan (from which the air is being sucked) should be replaced by a more powerful one, or, if there are two downcast shafts, the ventilation should be completely separated into two wings, building bulkheads with doors in the road joining the two wings; this layout is called a *sectionalized layout*.

The mutual effect of one fan on the other when two fans are working together in parallel is not seen only in the suction from one fan out of the other. Below are given two examples of joint operation: (a) of two radial-flow fans, and (b) of two axial-flow fans, showing how an increase in the resistance of a section of one fan is reflected in the whole air flow passing into the mine, and in the flow Q_1 from the second fan. The total air flow diminishes (Fig 12-10a, b, upper curve, Q_t); the flow from the second fan also increases, but this increase is larger with radial-flow fans (Fig. 12-10a, middle curve Q_1) than with axial-flow fans (Fig. 12-10b).

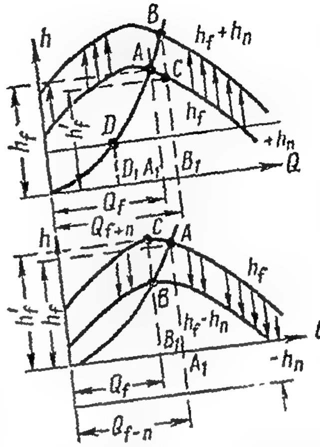

Fig. 12-11 Drawing up an overall characteristic for a fan working with a natural ventilating pressure

The equivalent orifice of a mine ventilated by two fans on the boundaries of the take is determined as follows.

Let Q_1 and Q_2 be the air flows, and h_1 and h_2 be the heads of the fans. The work done per unit time by the air in these splits will be equal to Q_1h_1 and Q_2h_2. Let us change the two fans for a single fan with a total flow rate of $Q_1 + Q_2$ and a head of h, the work done by the air evidently remains the same and therefore can be written

$$Q_1h_1 + Q_2h_2 = (Q_1 + Q_2)h$$

Determining from this equation the value of the head h and substituting it in Equation (7-30) for A and substituting $Q_1 + Q_2$ for Q we obtain

$$A = 0.38\frac{Q_1+Q_2}{\sqrt{\dfrac{Q_1h_1+Q_2h_2}{Q_1+Q_2}}} \qquad (12\text{-}1)$$

or, in the general form

$$A_t = 0.38\frac{[Q]}{\sqrt{\dfrac{[Qh]}{[Q]}}}$$

in which the square brackets indicate the action of summation.

When the air distribution from several fans operating jointly is the same as the natural distribution, that is, the air flow is inversely proportional to the square root of the resistances of the splits, the total equivalent orifice will be equal to the sum of the separate orifices calculated for each fan. For other distributions which require the expenditure of some additional work on overcoming the resistance of regulators, the total equivalent orifice will be less than the sum $A_1 + A_2$ The ratio $(A_1 + A_2)$ to A_t will be termed the *ventilation adjustment coefficient*, let us call it k_{va}

$$k_{va} = \frac{A_1 + A_2}{A_t} \qquad (12\text{-}2)$$

The closer the value of the coefficient k_{va} to 1, the better is the quality of the ventilation layout from the point of view of correspondence between the air quantities flowing through the separate splits, and their resistances

Calculations of the coefficient k_{va} for 20 mines of the Karaganda Basin made by one of the authors in 1945 gave values of k_{va} for various mines between 1 0 and 1 7

In mines where k_{va} is close to unity, in the transition stage to natural ventilation (e g. in winter at a metal ore mine), the relationship between the air flows in the splits will be approximately the same as with the fans located at the boundaries of the take.

12-5. OPERATION OF THE MAIN FAN IN SERIES WITH AN UNDERGROUND BOOSTER FAN INSTALLED IN ONE OF THE SPLITS

When the ventilation of one or more splits is difficult because of their high resistance, and the main fan cannot give them enough air, one or more booster fans are installed This layout is used both in metal ore mines and in non-gassy coal mines

An analysis of the joint operation of a main surface fan with a booster fan leads to the following conclusions

Depending on the power of the booster fan, the result of its installation may be·

(1) the required distribution of the air between the splits;

(2) an excessive increase of the flow in one split at the expense of the others,

(3) a reduction or even a reversal of the air flow in the splits which have no booster.

In both of the last instances, it is necessary (a) to reduce, if possible, the resistance of the common sections, (b) to increase the resistance of the split containing the booster; (c) to reduce the rotational speed (rpm) of the booster or to increase that of the main fan.

12-6. JOINT OPERATION OF A FAN WITH THE NATURAL DRAUGHT

Since the effect of the natural draught together with the main fan can be likened to that of fans in series, the solution of the problem of the joint operation of a fan with the natural draught does not differ in principle from the problem, already considered in Section 12-2, of fans working in series. The only difference is that with the assumed constancy of the natural draught, when the composite characteristics h_f and h_n are plotted, *the latter is added to the former*

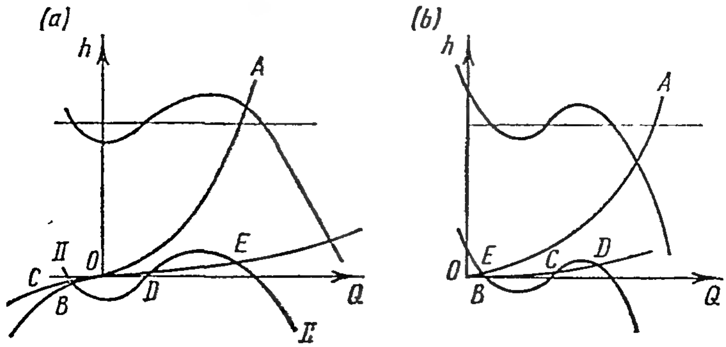

Fig. 12-12 Possible operating conditions for a fan with a natural draught.
(a) radial-flow, (b) axial-flow

as a constant intercept for all operating conditions of the fan. Naturally, if the natural draught opposes the fan then it *must be subtracted from the fan head.*

The diagram is shown in Fig 12-12. It shows that with a positive natural draught, or natural ventilating pressure (N.V.P.) the effective fan head diminishes, and with a negative N.V.P. it increases.

If a mine in broken country is ventilated through an adit and a vertical shaft having a low-pressure exhaust fan, then in summer the negative N.V P. will be considerable, with the following results: (a) during the operation of a radial-flow fan at a mine with a large equivalent orifice, the mine characteristic *OA* (Fig. 12-13*a*) will intersect the composite characteristic of the fan and the N.V.P. *II-II* at three points, *C*, *D*, and *E*; when the equivalent orifice (curve *OA*) diminishes, it will intersect only at point *B*. In the first instance two positive and one negative conditions are possible, in the second one negative, in the third quadrant, (b) in the operation of an axial-flow fan (Fig. 12-12*b*) with a large equivalent orifice,

three positive conditions are possible, at points B, C, and D; and with a small equivalent orifice a single positive condition at point E.

In considering the joint operation of a fan with the natural ventilating pressure in a mine having more than two outlets to the surface and located in broken country, it must be remembered that because the head developed by the fan decreases gradually with distance from the fan, being spent on overcoming the resistance of the airways, in some underground places connected with the surface, the residual head h_f of the fan may often be less than the N.V.P at the same point, h_n. Then, if the local N.V.P opposes the fan, the air flow will also oppose that caused by the fan.

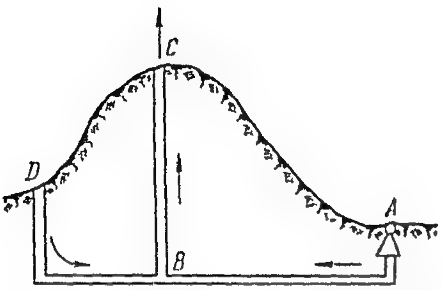

Fig 12-13 Layout for a fan working with a natural draught

For example, in the layout shown in Fig. 12-13, the N.V.P. in the section DB in winter will be into the mine, and, although the head developed by the fan at point A is considerably larger than the head in the section DB, this head, being largely spent on overcoming the resistance AB will not, at B, be able to overcome the effect of the reverse N.V.P. and the air, despite the forcing action of the fan, will enter the mine through the shaft collar D and may ice it up; the section DB will therefore be an intake section.

If the fan at point A is an exhaust fan, the converse will happen in summer, under the effect of the local N.V.P. the air flow BD will be upwards

To eliminate these reversals of the air flow, the resistance of the airway BC must be increased, e g. by installing a regulator in it, and correspondingly increasing the fan head.

CHAPTER 13

REGULATION OF THE AIR FLOW

In mine operation it is necessary:

(1) to change the air flow into the mine at constant equivalent orifice, e.g. when the gas emission changes;

(2) to maintain a constant air flow with a change in the equivalent orifice of the mine;

(3) to change the air flow simultaneously with changing equivalent orifice, e.g. with an increase in the production plan and the starting of new sections;

(4) in metal ore mines, to temporarily change the air quantity in the mine after blasting.

In all these instances the total air flow coming down the mine is changed.

However, there may be instances when a constant air flow into the mine has to be redistributed throughout the mine, e.g.:

(1) with changing air requirements in the different seams, ore veins, sections, etc.;

(2) with changes in the ventilation layout, e.g. with the change-over to a new horizon;

(3) during ventilation after blasting in metal ore mines for the rapid removal of the explosive fumes from stopes.

Some of these problems should be solved at the ventilation planning stage; the majority, however, are solved during the daily operation of the ventilation.

One feature of the adjustment of the air flow in coal and metal ore mines is the following.

In coal mines the total air flow is rarely changed, mainly during increases or reductions in output, or with increased gas emission; more frequently the air flow is redistributed between the different sections. e.g. when the quantity of gas is excessive in one of the sections of a gassy mine. In metal ore mines, on the contrary, several times per day the total air flow into the mine is changed as well as that to the various stopes, depending on the work done in them: drilling, blasting (for local or mass blasts), loading ore, or drawing it off

Observations of the distribution and regulation of the air flow through metal ore mines have shown that in some of them the air

reaches only the main haulage roads at the various horizons in adequate quantities and sweeps them well but then passes out to the upcast shaft, the sub-levels and the stopes receive very little air from the mine air flow.

The installation of auxiliary fans for the stopes and in the faces somewhat improves the air distribution, however, since these fans are usually not installed on the basis of calculations, the air flow is still not properly regulated In some metal ore mines the total capacity of these fans is larger than the power of the main fan, but even then it is not always possible to regulate the air flow into the stopes and ore bodies so that they all receive enough air. It is inevitable that the air then re-circulates, which is highly undesirable

13-1. METHODS OF REGULATING THE TOTAL AIR FLOW

Regulation of the air flow consists of. (1) changing the work of the fan; (2) changing the resistance against which the fan works, (3) a combination of methods 1 and 2

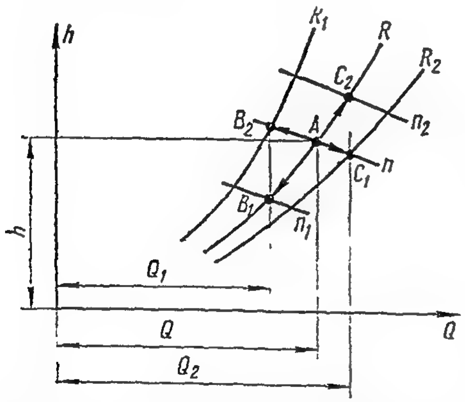

Fig 13-1 Regulation of the total air flow entering the mine

Let us compare the first two methods Let the mine fan give, at a rotational speed (rpm) n, an air flow Q, at a head h (Fig. 13-1, point A) To obtain the air flow Q_1 (where Q_1 is smaller than Q) we must reduce the rotational speed n to n_1 or increase the resistance from R to R_1; the points B_1 and B_2 will then be the new operating conditions of the fan

An increase of the air flow to Q_2 can be achieved by increasing the rotational speed of the fan to n_2 or by reducing R to R_2, at points C_2 and C_1. Finally, by changing R and n simultaneously, i e. using combined regulation, we will obtain intermediate operating conditions for the fan

Looking at Fig. 13-1, we can see that the conditions of points B_1 and C_1 are more favourable than B_2 and C_2, i e. for reduction of the air flow into the mine it is preferable to reduce the fan speed, rather than increasing the resistance by lowering a gate into the fan drift, and in order to increase the air flow it is preferable to reduce the resistance rather than increasing the fan speed.

For an accurate comparative evaluation of one or other method of adjustment, we must also allow for the changes in fan efficiency.

During regulation by changing the rotational speed of the fan, its efficiency remains unchanged, but when the output is varied by lowering or raising the gate, or by changing the equivalent orifice of the mine by removing the props, opening doors, or installing scale doors, the fan efficiency varies The variation depends on the position, along the characteristic, at which the fan was working before regulation began.

It must be stressed that even with a higher efficiency after regulation by changing the resistance, the fan power required will always be higher than when its rotational speed is changed. On the other hand, in regulation by increasing the equivalent orifice of the mine, the power demand even at a lower efficiency will always be less than when the output is increased by varying the rotational speed. This applies both to radial-flow and to axial-flow fans

When the fan delivers more air than the mine requires, it must be remembered that in radial-flow fans the power demand increases with increase of output, but with axial-flow fans working on the downward arm of the characteristic it diminishes. It is therefore preferable to reduce the output of a radial-flow fan by lowering a gate rather than to work it without regulation But with axial-flow fans there is no need always to resort to the use of a gate unless the reduction of air flow is dictated by other considerations, for example, excessive air velocity in the face

The operating conditions of the fan can be changed by the following methods.

A *Change in the rotational speed of the fan rotor.*

(1) by reduction gearing or stepped pulleys, involving regulation by steps which is now rare,

(2) by a hydraulic coupling giving smooth regulation, used when the amount of regulation required is not excessive;

(3) other methods, such as special cascade equipment, rarely if ever used in the USSR.

B. *At constant speed of the fan rotor*

(1) with axial-flow fans, varying the angle of attack of the rotor blades;

(2) varying the angle of the guide vanes either in an axial-flow or a radial-flow fan. This can be done both by remote control and automatically

In the particular case of metal mines, where the fan output must be increased after blasting, the following may be used (a) hydraulic couplings, (b) two fans of different outputs; (c) two fans together; (d) the use of one two-speed motor or of two motors of different speed, on the same shaft

Variation of the fan output by varying the resistance is usually achieved by a gate in the fan drift; as pointed out above, this method

is uneconomical A much greater effect is achieved by reducing the mine resistance by removing the props or widening the airways, cleaning up fallen rock, changing the supports for a smoother type, or by rebuilding the fan drift or air crossings, or shortening the air circuits, or dividing the air into the largest possible number of splits

It is sometimes possible to increase the air flow into the mine by reducing the resistance of the fan installation and by reducing the short circuits at the surface.

In conclusion we must emphasize that any increase of the fan speed must be preceded by an inspection of the rotor *to find out how far it is worn and whether the proposed fan speed will be dangerous* Also, to avoid overloading the motor of a radial-flow fan when the fan output is increased, the power required from the fan must be calculated and compared with the power of the existing motor.

13-2. CHANGING THE AIR QUANTITIES FLOWING THROUGH THE SPLITS

Courses on mine ventilation generally treat the regulation of the air flow as a matter of redistributing the flow, increasing it in some splits at the expense of others. This redistribution is usually achieved, as explained below, by installing a scale door But in practice the problem often arises of increasing some flows without appreciably reducing others This problem is solved by installing booster fans in the splits which require increasing

In this connection we shall consider the following

(1) regulation of the air flow by building a scale door containing a regulator (so-called negative regulation),

(2) regulation by reducing the resistance to the air flow in the split which must be increased (so-called positive regulation),

(3) increasing the flow in some splits by installing booster fans in them,

(4) regulation by a combination of methods.

13-2 1 Regulation of the Air Flow by a Regulator (Negative Regulation)

Let us consider the simplest instance of two splits, ABC and ADC, having resistances R_1 and R_2 (Fig 13-2). With the natural distribution of the air flows Q_1 and Q_2 they will be in the proportions indicated by Equation (9-20) Let us assume that the ratio Q_1/Q_2 is unsatisfactory, and that we wish to reduce the flow in ADC from Q_2 to Q_2', thereby increasing the air flow through ABC from Q_1 to Q_1' and obtaining a different value of the ratio

$$m=\frac{Q_1'}{Q_2'}>\frac{Q_1}{Q_2}$$

One of the ways of solving this problem is to build a scale door, containing a regulator, in the split which has too much air. The dimensions of the regulator are found in the following way:

Stage 1. Let us call the resistance of the regulator R_{reg}. The total resistance of the airway ADC after the installation of the regulator will be $R_2 + R_{reg}$. Since the heads of the two splits must be the same,

$$h_{ABC} = R_1 Q_1'^2 = h_{ADC} = (R_2 + R_{reg}) Q_2'^2$$

and the resistance of the regulator can be found from the equation

$$R_{reg} = R_1 \left(\frac{Q_1'}{Q_2'}\right)^2 - R_2 = R_1 m^2 - R_2 \qquad (13\text{-}1)$$

Our problem consequently reduces to finding such dimensions of the regulator which will bring its resistance to

$$R_1 m^2 - R_2$$

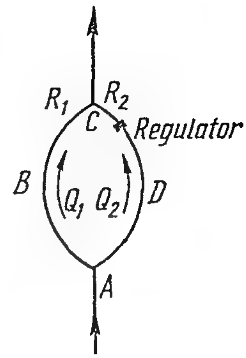

Fig 13-2 Diagram for calculating the size of a regulator

Stage 2. With the equation obtained in Chapter 8, for local resistances, $R_1 = 0\ 0612 \frac{\xi}{S^2}$ we obtain the coefficient of resistance

$$\xi = \frac{1}{0\ 0612} R_{reg} S^2 = 16.3\ R_{reg} S^2$$

$\left(\text{more accurately, } \xi_{reg} = \frac{2g}{\gamma} R_{reg} S^2\right)$

in which S = the cross-sectional area of the airway at the regulator. Because any regulator is a local resistance of thin plate type, for which the coefficients were experimentally determined long ago and are discussed in all hydraulics courses, we shall use these coefficients to construct the diagrams (Fig. 13-3) in which the ratio $\frac{s}{S}$ is plotted along the abscissa axis (s being the area of the regulator), and the coefficient ξ is plotted along the ordinate axis. Because of the wide variations in the value of ξ, and since we do not wish to use logarithmic co-ordinates, we give three curves, plotted to three different scales.

Let the resistances R_1 and R_2 be 60 and 40 murgs, and the area $S=4$ sq m It is required that the first airway should receive 60 per cent of the air passing through the splits Determine $m = \frac{60}{40} = 1\ 5$, from Equation (13-1) the resistance of the regulator will be

$$R_{reg} = 0\ 060 \times 1.5^2 - 0.040 = 0.095 \text{ kilomurg}$$

The coefficient of resistance of the regulator will be

$$\xi_{reg} = 16.3 \times 0.095 \times 4^2 = 24.8$$

From the graph, for this coefficient the value $s/S = 0.27$, consequently $s = 0.27 \times 4 = 1.08$ sq m

Considering the sudden widening of the air flow downstream of the regulator as a shock, it is easy to obtain the following expression for calculating the area of the regulator.

$$s = \frac{S}{0.65 + 2.63S\sqrt{R_{reg}}} \text{ sq m} \tag{13-2}$$

where S = cross section of the airway at the regulator, m²
R_{reg} = resistance of the regulator, kilomurgs.

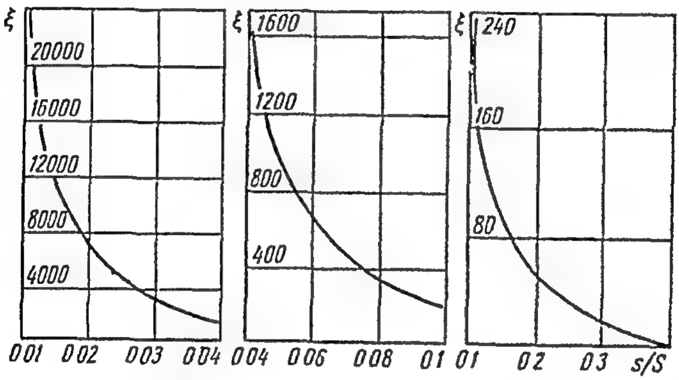

Fig 13-3 Graphs for the coefficient of resistance, ξ, of a regulator in relation to the ratios of the areas of the regulator, s, and of the airway, S

From this formula, for the example under consideration,

$$s = \frac{4}{0.65 + 2.63 \times 4\sqrt{R_{reg}}} = 1.03 \text{ m}^2$$

Another formula, used when s/S is more than 0.5, is

$$s = \frac{S}{1 + 2.38S\sqrt{R_{reg}}} \tag{13-3}$$

If, instead of two splits, we had three or more, the problem would be solved as follows.

We calculate the heads of all the splits

$$h_i = Rp_i^2$$

in which p_i is the air flow, per cent, passing through each split; a regulator should evidently be installed in every split, except the one with the highest head.

We determine the values

$$m_i = \frac{p}{p_i}$$

in which p_i is the air flow, per cent, in the split which does not require a regulator, consequently

$$R_{reg} = Rm_i^2 - R_i$$

in which R is the resistance of the split without a regulator.

The rest of the problem is solved as above.

The pressure drops calculated from the formula

$$h = \xi_{reg} \frac{v^2}{2g} \gamma$$

on which s/S is more than 0.13, agree with direct measurements of heads of underground regulators with an accuracy which is adequate for practical purposes. With $s/S = 0.13$ the values obtained for s should be increased by 7.5 per cent, and with $s/S = 0.1$ they should be increased by 12.5 per cent.

Stage 3. After determining the resistances of the regulators we find the total resistance of the system by the rules indicated in Chapter 9.

Stage 4 The characteristic of the ventilation circuit is drawn, and its point of intersection with the characteristic of the fan installed at the mine is found. The abscissa of this point will be the air quantity which the fan must supply after the regulators have been installed.

Stage 5. The total air flow is distributed among the splits by the rules stated in Chapter 9.

The solution of this problem may give the following results: (a) the air flow through all the splits will be adequate; (b) the air flow through the split which required more air becomes adequate, but the flow through the others becomes inadequate; (c) the air flow through all the splits is inadequate.

In the two last instances it is evidently impossible to obtain the required air flows or ratios of air flows between the splits by installing regulators But the problem can be solved, for example, by installing a booster fan in the split which needs more air.

Because any regulator increases the resistance of the split in which it is installed, and consequently the resistance of the mine as a whole, we can make the following extremely important statement: *Any regulator reduces to some degree the air flow into the mine.* Secondly (this results from the first statement), *in a redistribution of the air flow by a regulator, the gain in air flow in one split is always less than the air flow lost in the other split.*

These two important statements force us, in the regulation of air flow, to reduce to the minimum the number of regulators, and sometimes to do without them altogether, neglecting any slight disparity which there may be between the actual air flow to the working places and their air requirements

Generally speaking, the change in the ventilation of the mine and of the splits after installing regulators depends on the relative resistances of the elements of the network, on the locations of the regulators in the splits, and on whether the fan characteristic is steep or gently sloping.

Therefore the following indications can be given concerning the changes in the ventilation of the mine or its splits:

(1) The reduction of the fan flow rate will be the less, and consequently the effect from the installation of the scale door will be the larger, (a) the steeper the right-hand (working) branch of the fan characteristic (therefore an axial-flow fan after regulation by a door will give more air than a radial-flow fan); (b) the larger the number of splits in the mine ventilation network, and the less the importance of the airways in which the flow rates must be redistributed; (c) the larger the resistance of the whole network by comparison with the resistance of the splits

(2) The installation of a scale door should give the larger increase of flow rate in the split which requires more air, the smaller the resistance of this split by comparison with that containing the scale door

Generally, to regulate the air flow by a scale door, it is necessary. (a) first to assume the resistance of the regulator to be infinity; (b) to estimate then the total resistance R_t of the whole mine with the fan installation; (c) to determine Q_t from the fan characteristic; (d) to determine Q in the split which needs more air; this will be the maximum air flow which can be obtained in this split by building the scale door in the other split.

If these calculations are performed for each of the two splits, we obtain the limiting (maximum) air flows which can pass in each split when a scale door is installed in the next split. An even more visual picture of the interaction of the changing air flows in the two splits can be obtained from a diagram on which are plotted the air flows in each split at various openings of the regulator.

For example, Fig. 13-4 shows a system consisting of two splits with the resistance of the fan installation, $R_f = 80$ murgs; $R_{AB} = R_{EF} = 10$ murgs; $R_{BCE} = R_{BDE} = 400$ murgs; the radial-flow fan without shock absorber has an impeller, diameter of 1.2 m, and the air flow rate u is 30 m/sec Let us build a scale door in split BDE and vary its dimensions and consequently its resistance; the air flowing in the splits will change (calculations not shown) as

indicated on the curves of Fig. 13-5 An inspection of these curves shows that·

(1) the installation of a scale door in one of the splits results in. (a) a comparatively small increase in air flow in one of the splits (in the given case, the maximum increase is 40 per cent), (b) a considerably greater reduction (in absolute and relative value) to the air flow in the other split, and (c) a reduction of the total air flow,

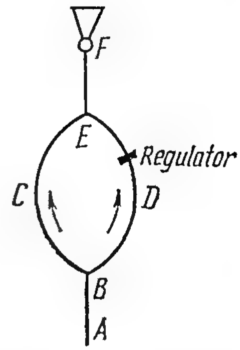

Fig. 13-4 Diagram of two splits with a regulator in one of them

(2) the main redistribution of the air flow between the splits is achieved even with a comparatively small resistance of the regulator; for example, with $R_{reg} = 5{,}000$ murgs, the air flow in the split already increases by 30 per cent; further increases of the resistance of the regulator up to 40,000 murgs increase the air flow in this split by only 4 5 per cent, at the same time, the reduction of the air flow in the parallel split reaches 65 per cent at $R_{reg} = 5{,}000$ murgs, and about 86 per cent at $R_{reg} = 40{,}000$ murgs.

Conclusion. *after achieving a certain level of resistance of the regulator, further increase of its resistance gives a very small gain in air flow in the split requiring more air, but considerably reduces the flow in the regulated split.*

By building a scale door in a split, only those conditions of ventilation can be obtained which correspond to the curves a_1' and b

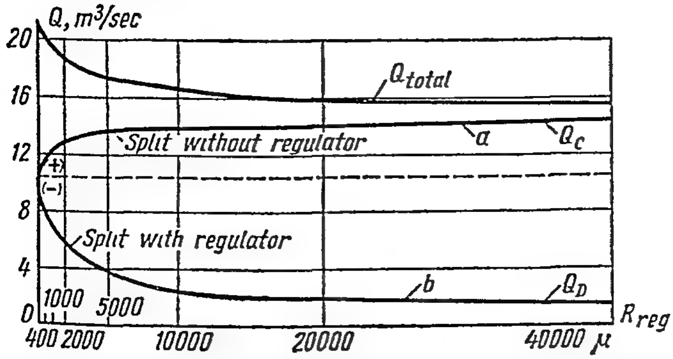

Fig 13-5 Change in the air flow passing through two splits, in relation to the resistance of the regulator

(Fig 13-5) For example, the condition $Q_C = 15$ m³/sec and $Q_D = 5$ m³/sec will not be achieved from this fan with any dimensions of regulator, although the sum of the air flows $Q_C + Q_D = 20$ m³/sec is less than the original air flow of 21.1 m³/sec.

Changing a radial-flow fan for an axial-flow fan, i.e. a fan with a steeper working sector on the characteristic, would enable the required relation between the air flows in the splits to be achieved with a smaller absolute reduction in the flow in the regulated split; the installation of the regulator, as pointed out above, would result in a smaller reduction of the air flow from the axial-flow fan by comparison with the radial-flow fan.

When the erection of a scale door with regulator in the nearest split does not produce the necessary effect, and the air flow must absolutely be increased as quickly as possible, it is possible to resort to the installation of a regulator in a larger split nearer the shaft, however, the required improvement in this split will be achieved at the cost of a significant, at times, deterioration in the ventilation of the remaining splits.

13-2 2 Regulation of the Air Flow by Reducing the Resistance of the Split Which Needs More Air (Positive Regulation)

As pointed out above, regulation of the air flow by building a scale door with regulator sometimes does not yield the desired effect or it excessively disturbs the air flow in neighbouring splits.

It may be much better to reduce the resistance of the split which requires more air And although this reduction is usually difficult, involving laborious and consequently expensive work in removing the props, cleaning and widening the existing workings and those being driven, this is often the only correct action.

For example, let the air flow passing in one of the splits be

$$Q_1 = \frac{Q_t}{\sqrt{\frac{R_1}{R_2}}+1} \tag{a}$$

where R_1 and R_2 = resistances of the splits

Q_t = total air flow.

It follows from Equation (a) that an increase of Q_1 to Q_1' can be achieved by reducing R_1 to R_1' and the value of R_1' can be obtained from the equation

$$R_1' = \left(\frac{Q_t}{Q_1'} - 1\right)^2 \cdot R_2 \tag{b}$$

Thus, if the ratio of the total air flow to that required in one of the splits is known, as well as the resistance R_2 of the other split, then the required resistance of the first split is obtained from Equation (b), and its cross-sectional area, from the equation

$$R_1' = \alpha LP/S^3 = \frac{\alpha L 4\ 16\sqrt{S}}{S^3} = 4\ 16\alpha L S^{-2\ 5}$$

from which

$$S^{2.5} = \frac{4.16\alpha L}{R_1'} \tag{13-4}$$

Unlike the method of regulation by a scale door, regulation by increasing the dimensions of the airway does not diminish, but increases the overall air flow coming underground; this method is therefore recommended for mines with neglected workings having small equivalent orifices

In regulation by reducing the resistance of one of the splits it is possible to achieve a greater increase of the air flow in it than by regulation with a scale door, and this increase is achieved at the expense of a considerably smaller reduction of the flow through the next split Also the increase of the flow in the first split exceeds the reduction in the other split.

13-2 3 Increase of the Air Flow by Installing a Booster Fan in the Split Which Needs More Air

Instead of installing a scale door or reducing the resistance of the airway, another method used for increasing the air flow underground in a difficult split is to install a booster fan in it. Unlike auxiliary fans, these fans pass through them all the air which ventilates the particular district or wing of the mine, and they work without ventilation ducting.

The installation of underground booster fans is allowed in coal mines and metal mines with permission from the appropriate authorities.

Booster fans often greatly increase the ventilation; their installation is advisable when, for example, the increase of air flow is difficult or impossible to achieve either by building a scale door in neighbouring splits, or by widening the airway (because of the extremely high cost of widening), or by increasing the fan speed or by installing a new main mine fan with a larger peripheral speed On the other hand, booster fans possess the following significant disadvantages.

(1) They often involve the re-circulation of a considerable part of the air underground; re-circulation is dangerous in mines working seams subject to spontaneous combustion, as well as in gassy mines, because in the first it may cause underground fires, and in the second it may result in the district filling with gas; in metal mines the time required to clear the blasting fumes from the stopes is longer.

(2) The inspection of booster fans underground is often less strict than that of surface fans, therefore they are often out of action for

various reasons, in gassy mines a dangerous accumulation of methane may occur during a stoppage of a booster fan, if the gas fills back to the fan, the re-starting of the fan may be extremely dangerous (a number of explosions have resulted from this)

(3) Booster fans are often installed without either calculation, or check on their joint operation with the main fan, sometimes their output exceeds that of the main fan, when this happens the booster fan can give results greatly differing from those desired, and may even impair the ventilation.

(4) When a mine fire or explosion occurs, the booster fan is either destroyed, or becomes inaccessible, and it is therefore difficult to restore the ventilation in its district

For the safe operation of booster fans the following are necessary. (1) the operating conditions should be the same as for main fans on the surface (particularly for gassy mines); in non-gassy mines or metal mines some relaxation of these conditions may be allowed; starting and stopping of the fans must be strictly in accordance with the regulations, and with authority from the ventilation engineer, (2) they must be installed in places where re-circulation is impossible (page 378), (3) a booster fan must be installed in an intake airway.

In addition, it is desirable to control booster fans remotely In gassy mines when the fan stops, the power should be automatically switched off from the district served by the fan, and the doors of the air lock beside the fan should be opened.

Booster fans are installed in individual splits of closed or open ventilation systems As regards the methods of operation, they are classified as (a) fans working through a stopping, and (*b*) fans working without a stopping

Increase of the air flow by installing a booster fan built into a stopping Fig 13-6 shows (for the same layout as in Fig. 13-4) how the air flow will change in the splits when a booster fan is installed in one of them, along the abscissa axis is plotted the fan speed, along the ordinate axis the air flow. Comparing this drawing with Fig. 13-5 we can draw the following conclusions (1) the total air flow increases instead of diminishing; (2) the air flow in the split which needs more air can be increased very much more than by regulation with a scale door, (3) the quantity of air passing through the other split, although it decreases, does so very much less than by regulation with a scale door

As in regulation by a scale door, not all operating conditions can be obtained by installing a booster fan, but only those which are indicated (for this example) on curves *a*, *b* (Fig 13-6), furthermore, to each air flow in one of the splits (for example, that with the fan) there corresponds a fully determinate air flow in the other split

(without the fan), which is obtained from the ordinate passing through the curve *a* and intersecting the curve *b*.

Analysis of the problem of regulation by a booster fan brings out one more important conclusion; whatever booster fan is installed for regulation, a change in the air flow will always follow the same curve, in this instance *a* or *b*; it will not be possible with different booster fans to achieve other conditions, not on these curves, which, as pointed out, are linked together, the difference in operation of

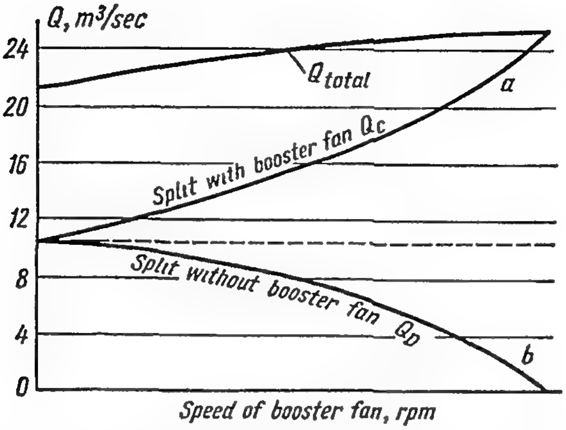

Fig 13-6 Change of the air flow in two splits, with a booster fan in one of them

the different fans will merely be that more powerful fans will create along the curves *a* and *b* conditions further to the right than less powerful fans

In each individual instance, before choosing one or another method of regulating the air flow, the results and costs must be compared. Usually for regulating within a district or a stope, the first method (scale door) is used, for redistributing the air between groups of districts or seams, more elaborate regulation can be achieved by the two other methods

In order to avoid unsuitable operating conditions with a booster fan, a careful check must be made before installation to see how its operation will affect the other splits If its installation appears unacceptable, it may be necessary to abandon the attempts to provide the required air flow in the split which needs more air, or to resort to a combination of different methods of regulation

A pictorial comparison of all three methods of regulation is shown in Fig. 13-7, on which the abscissa axis carries the percentage increases of air flow in the split which needs more air, the ordinates being percentage reductions in the split being decreased. As the diagram

(Chapter 7, Section 2); part of this head is lost by an air shock as the air suddenly expands, the remainder being utilized; however, because of the ejector effect an additional quantity of air is sucked along the airway beside the fan

For all such fans (they are sometimes called pushers) there are fully determinate conditions in which more air is provided without a stopping and others in which more is provided with a stopping, in each individual instance, if the air flow provided by the fan is measured with an open and with a closed door, it can be shown which of these quantities is the greater; generally speaking, the smaller the resistance of the network against which the fan is working, the more likely it is that operation without a stopping will be more effective.

The mining engineer T.E. Gumenyuk proposed the following equation for calculating the air flow through an airway during the operation of a fan without a stopping.

$$Q_t = Q_f \sqrt{\frac{S}{s_f\,(8\ 2RS^2+1)}} \tag{13-5}$$

in which Q_f = air flow provided by the fan
S = cross-sectional area of the airway in which the fan is installed, m^2
s_f = cross section of the outlet of the fan or of the short ducting connected to it, m^2
R = resistance of the network in which the fan works

T E Gumenyuk also proposed the following equation for the resistance of the network, below which operation without a stopping would be effective:

$$R = \frac{0\ 122}{S}\left(\frac{1}{s_f} - \frac{1}{S}\right) \tag{13-6}$$

If the resistance of the network is less than that obtained from Equation (13-6), operation without a stopping is better than operation through a stopping.

To obtain the air flow Q_t which will be given by a fan working without a stopping the following procedure is used

(1) The overall characteristic of the fan is drawn.

(2) On the same drawing the characteristic of the resistance of the outlet from the fan is plotted in the form of a parabola, passing through the origin of the co-ordinates, this resistance is equal to

$$R_{out} = 0.0612\xi_{se}\,\frac{1}{s_f^2} \tag{13-7}$$

in which s_f has the same value as before, and the value of the coefficient ξ_{sc} of local resistance for the sudden expansion at the outlet

of the air can be obtained from the formula proposed by A.A. Kharev,

$$\xi_{rough} = \xi_{smooth}\left(1 + \frac{\alpha}{0.0010}\right) \tag{13-8}$$

In this equation α is the coefficient of friction of the working and ξ_{smooth} is the coefficient of local resistance for the sudden expansion which, according to I. E. Idelchik, for various values of s_f/S is equal to

s_f/S	0.0	0.1	0.2	0.3
ξ_{smooth}	0.95	0.85	0.68	0.52

The abscissa of the intersection of the characteristics of the fan and the parabola drawn for R_{out} is the fan output.

(3) Equation (13-5) is used for calculating the air flow through the network.

If this quantity is less than required, the calculation is repeated with a fan of larger diameter, or several fans are used.

Note: The overall characteristic of the fan is found in the following way: from the static characteristic of the fan a number of values of the air flow Q and the head h are extracted. For each condition the velocity head at the outlet from the fan is calculated from the equation

$$h = \frac{\left(\frac{Q}{s_f}\right)^2}{2g}\gamma \text{ mm } H_2O$$

in which γ is the specific gravity. This head is added to the fan head, a line drawn through these points is the overall characteristic which should be extrapolated approximately in the region of low pressures.

The air flow given by the fan without a stopping is affected by the type of fan and the shape of the ducting connected to its casing or outlet, i.e. whether the ducting is short, gently widening (like diffuser), or narrowing (nozzle type) or of constant diameter; investigations have shown that the greatest increase of air flow is given by axial-flow fans with a slightly restricting fitting on the outlet; the gain in air flow is here achieved by increased velocity head at the outlet from the ducting (notwithstanding the large "shock" loss caused by the sudden widening of the flow). The installation of such nozzles on radial-flow fans does not increase their output.

13-2.4 Combined Methods of Regulating Air Flow

Notwithstanding the considerable widening of the range of regulation obtainable by a booster fan in one of the splits, there can be instances when the required conditions are obtainable neither

by building a scale door with a regulator nor by the use of a booster fan.*

Sometimes the range of possible conditions can be widened by installing a booster fan in one split and building a scale door in the other; in other cases, these measures will be inadequate and one of the following may also be needed (a) increasing the speed of the main fan or changing the main fan; (b) putting a booster fan into the second split; (c) installing a second booster fan in the second split, (d) reducing the resistance of the common section

13-2 5 Regulation of the Air Flow in Compound Networks

The regulation of the air flow in a compound network differs from regulation in a simple split system in that the required relation between the air flows can be achieved by installing the scale door in various splits and not in any particular one; in practice, as well as in ventilation planning, the need to calculate regulators for compound networks is comparatively rare.

This need may arise in investigating the ventilating conditions of a mine, and in working out measures to improve them; in this instance, the most suitable solution can be to make a pressure survey of the mine and subsequently, after working out the results in the office, to apply them to an electrical analogue for investigating mine networks These instruments can also be used for investigating other methods of regulation.

13-2 6 Regulation by an Air Curtain

A new method of regulating air flow, which seems to hold some promise, is regulation with the aid of air curtains, proposed by the mining engineer Tsoi. It consists of a distributor in the form of a vertical slit placed beside the airway and delivering an opposing plane stream of air at some angle to the main air flow, the action of this curtain on the main air flow is to retard it and to direct part of it into a split. By changing either the angle of inclination of the flow in relation to the axis of the airway or the air flow passing out of the slit, it is possible to regulate the air flow in the two splits within wide limits. To create the air curtain, an auxiliary fan is installed not far away, connected to the curtain by a length of ducting not less than 300 mm diameter The slit may be 50 to 100 mm wide The

* It must not be forgotten that the problem is to redistribute a flow already provided from an existing main fan, and not to provide stated operating conditions This latter problem can be solved for any air quantities flowing through splits (see Part Three)

distributor may also be placed on both sides of the airway, in this case the angle of incidence of the stream is 45 to 60 degrees.

An advantage of this method of regulating air flow is the absence of any doors; consequently, transport is simplified and there is no need for a door trapper

The quantity of air flowing through the slit is calculated from the equation below

$$q_0 = \frac{Q_0}{\sqrt{\frac{B_0 q_1'}{b_0 q_2'}(a\eta^{-b}+c)}} \tag{a}$$

in which q_0 = air flow passing through the slit
Q_0 = total air flow passing into the split
B_0 = width of the working
b_0 = width of the slit of the air curtain
a, b, c = constants whose values are given below in relation to the angle at which the slit impinges upon the working

Inclination, degrees	a	b	c	
0	0 379	1 83	1 25	
30	0 571	2 00	2 33	
45	0 276	2 97	2 41	most favourable angles of incidence
60	0 385	2 37	2 33	

q_1' and q_2' = natural air flows passing through the "curtained" and the other air split

η = efficiency of the curtain, equal to $\frac{q_1'-q_1}{q_1'}$, where q_1' and q_1 are the air flows passing through the "curtained" split before and after installation of the curtain.

Note: The air flow obtained from Equation (a) should be multiplied by the correction factor 1 3

Another method similar to that described consists in installing, in the working in which the air flow must be diminished, a fan without a stopping, to oppose the flow through it.

CHAPTER 14

AIR LEAKAGES

14-1. GENERAL

One feature of underground ventilation is the presence of large leakages along the airways. These leakages include short circuits at the surface through badly fitting doors of the upcast shaft and through subsiding ground, cracks, and old workings or the bottoms of quarries. Leakages and short circuits are a serious disadvantage to the operation of ventilating plant, because of the inadequate air supply, dangerous accumulations of explosive gases occur, there is an increase of the time required to ventilate working places after blasting, dust settles where it is formed, and working conditions become unsatisfactory; all this naturally influences the output, lowering it. Losses of air through the worked-out area and cracks in the mineral can cause spontaneous fires. Finally, losses cause a completely useless expenditure of power in the supply of extra air to compensate for the leakages and short circuits

Numerous investigations of air leakages in the mines of the Donets Basin in the last few years have shown that in most of the mines investigated the losses are extremely large, reaching 50 per cent or more of the quantity of air flow Q which passes down the mine. It may be supposed that in other coalfields and metal mines the air losses are also considerable.

14-2. METHODS OF DETERMINING AIR LOSSES

The following methods are used for determining air losses·

(1) Fairly large losses can be directly measured by an anemometer

(2) Leakages are determined by the differences between the quantities of air flow Q_1 and Q_2 before and after the ventilating structure, for example, a stopping through which the air is leaking

(3) In the airway along which the leakage is taking place, a stopping with a small regulator is built, and the air velocity and the quantity of air flow through it are measured; since the resistance of this regulator will be considerably less than that of the ventilating structure through which the air is leaking, its presence does not affect the leakage quantity

As a qualitative indication, leakages can be measured by smoke: the stream of smoke will show the location of the leaks A fairly stable smoke is obtained from titanium oxide

Air leakages can also be investigated by the use of radioactive isotopes after they have been released into the intake air and found again in the return air, the paths of air movement through the waste, packs, stoppings, etc , can be established

14-3. THE LAW OF RESISTANCE OF AIR LEAKAGES

Judging by the results of air leakages in laboratory conditions, the movement of air through a layer of fine sand, stone dust, etc , is laminar The leakage of air through tight packs or strip packs or falls of rock after some months of consolidation is also subject to the same law of laminar flow or to the one close to it In all other cases, as was shown in careful investigations made by M. A Patrushev in 1955, the resistance will be intermediate or quadratic. Observations on air leakages through ventilating structures (mainly through stoppings) have shown that

(a) the main mass of the air is sucked through the perimeter of the stopping where it touches the walls (80-90 per cent in solid ground);

(b) the law of air flow through stoppings of all types in their original condition is generally quadratic; if the surface of the stopping is plastered with clay, this hardly affects the leakages at all; complete plastering of the surface and of the perimeter with clay reduces the exponent x of Q, on an average, down to 1 78 and to 1 65 when cement mortar is used for plastering Extremely careful and complete plastering reduces the exponent x to 1 6 and 1 5, respectively. However, six months after the careful plastering, the leakages again become turbulent

Leakages through lump material in bunkers follow an intermediate law, through a worked-out area, near the face, they are quadratic, through the joints in ventilation ducting they may be considered quadratic

The Coefficient of Permeability to Air. Equation (8-16) can be written in general form as follows

$$h = RQ^x$$

in which the exponent x varies from 1 to 2, but

$$Q = vS$$

consequently

$$h = Rv^x S^x$$

This equation is also valid for leakages of air through any material such as a stopping or a wall. Let us consider an air flow through a prism of a cross section of 1 m² and length l, metres, and the equation of movement will then be rewritten·

$$h = rl \times 1^x v^x$$

where r is the resistance of a prism 1 metre long.

The reciprocal of this resistance, i. e.

$$\frac{1}{r} = \frac{v^x}{h/l}$$

is called the coefficient of permeability to air and is denoted by the letter k. With laminar, turbulent, and intermediate (transitional) laws of flow, the coefficient of permeability is equal respectively to:

$$k_{lam} = \frac{v}{h/l}; \quad k_{tur} = \frac{v^2}{h/l}; \quad k_{tr} = \frac{v^x}{h/l} \tag{14-1}$$

The dimensions of this coefficient are

$$[k_{lam}] = \left[\frac{\text{m}^4}{\text{sec kg}}\right]; \quad [k_{tur}] = \left[\frac{\text{m}^5}{\text{sec}^2\ \text{kg}}\right]; \quad [k_{tr}] = \left[\frac{\text{m}^{x+3}}{\text{sec}^x \cdot \text{kg}}\right]$$

Let us multiply and divide the equation for k_{tur} by S^2, after substituting Q^2 for v^2S^2, we obtain:

$$k_{tur} = \frac{Q^2}{S^2\ h/l} \tag{14-2}$$

For the case of air leakage through a stopping, $h = R_{st}Q^2$ (see page 280), we obtain

$$R_{st} = \frac{R_{tur}}{S^2}$$

in which R_{tur} is the resistance of 1 m² of stopping.

Substituting this value in (14-2) we obtain

$$k_{tur} = \frac{Q^2}{\frac{R_{tur}Q^2}{S^2} \cdot \frac{S^2}{l}} = \frac{l}{R_{tur}} \tag{14-3}$$

and consequently for laminar flow.

$$k_{lam} = \frac{l}{R_{lam}} \tag{14-4}$$

M.A. Patrushev on the basis that a stopping leaks mainly along its perimeter, proposed that the coefficient of permeability to air should be equal to the quantity of air in m³/sec passing through each running metre of perimeter of a stopping 1 m thick, under a pressure of 1 mm.

14-4. THE AIRTIGHTNESS OF VENTILATION STRUCTURES

The most suitable measure of the airtightness of a ventilating structure is its resistance; however, for calculations (see page 375) it is more convenient to deal with the absolute quantity of air leaking through the structure We should also point out what percentage of air is lost in passing through a leaky structure. It is proposed that the quality of doors, stopping, etc , be estimated on the basis of the following figures

very good if $p \leqslant 5\%$
good if $10\% \geqslant p \geqslant 5\%$
average if $20\% \geqslant p \geqslant 10\%$
bad if $p = 30\text{-}40\%$
very bad if $p > 50\%$

14-5. CLASSIFICATION OF LEAKAGES

1 Classification by location of the leak is as follows

(a) leakages or short circuits through the collar of the upcast shaft,

(b) leakages in the pit bottom, including those through the coal bunker when the coal is hoisted by skips,

(c) leakages through doors and other ventilation structures underground,

(d) leakages from one airway to another driven parallel with it;

(e) leakages through the worked-out area

2 Classification according to the type of leakage is·

(a) local leakages (including the first three types above);

(b) uniformly distributed leakages

The reduction of local leakages is usually much easier and simpler than the reduction of leakages through the worked-out area and between parallel airways

14-6. LOCAL LOSSES OF AIR

A local loss is one which is confined to a particular place in the network of mine workings Let us consider local losses by starting from the collar of the downcast shaft up to the fan, not distinguishing between losses inwards and losses outwards

(a) *Downcast shaft* Generally speaking, leakages here should be zero, however, when the shafts are connected at several horizons, short circuits of air through stoppings and doors which are not airtight may be very considerable

(b) *The pit bottom and its surroundings* If the pit bottoms of the hoisting shaft and the upcast shaft (placed close together in the middle of the mine take) are at the same horizon and connected by a drift with doors, the loss through the doors often reaches 20 to 30 per cent because of the large pressure difference across them, to prevent this loss as well as that indicated under (a) there must be not less than two strong masonry stoppings in the drift, cut into the sides and roof and carefully fitted with doors.

Sometimes large air leakages occur through the loading bunkers used with skip hoisting; to reduce these losses the same procedure is used as in the buildings at the surface of the mine (see Chap. 13)

Normal losses can be considered to be about 5 per cent for cage hoisting and 10 per cent for skip hoisting.

(c) *Leakages through ventilation structures, stoppings, doors, crossings, etc* Investigations in the Donets coalfield, made by the department of ventilation of the Leningrad Mining Institute in 1953 and 1955, have shown that these losses in many mines reach 40 to 50 per cent of the air flow entering the mine.

Stoppings. The amount of leakage through stoppings depends on the pressure drop across them, the permeability to air of the stopping, its dimensions and thickness An investigation of stoppings showed that the air leaks mainly across the perimeter of the stopping where it touches the adjacent rocks, by plastering the stopping with clay, or preferably with cement grout, the leakages are reduced on the average by 40 per cent, and with very careful plastering, by 70-80 per cent; this plastering must be repeated every two months if clay is used and every six months if use is made of cement grout; if the wall rocks are cracked, the leakages rapidly increase.

Table 14-1 (according to M.A. Patrushev) shows the average values of resistance of stoppings of various types in solid rock and cracked rock with a cross section of $S = 5$ m^2, and thickness b, equal to 1 m, the air leakages under the quadratic law can be calculated from the equation.

$$Q = \sqrt{\frac{h}{R_{st}}} \text{ m}^3/\text{sec}$$

where h is the pressure drop, mm water

With other values of the area S and the thickness of the stopping b, its resistance changes in direct proportion to the thickness and in inverse proportion to the area of the stopping.

A typical leakage through a single stopping can be assumed to be not more than 20 m^3/min for a head of 50 mm or more, not more than 15 m^3/min for a head of 30 mm; and not more than 10 m^3/min for a head of 10 mm.

TABLE 14-1 Average Resistances of Perimeters of Stoppings (According to M A Patrushev)

Description		R_{st}, kilomurgs	
		solid rock	cracked rock
Slag or rubble concrete	In uncracked pillars	6,100	1,700
Stone	In uncracked pillars	4,600	1,500
Slag concrete blocks	In uncracked pillars	3,000	1,000
Round timber	In uncracked pillars	1,700	600
Plank	In uncracked pillars	900	—
Round timber in cracked pillars		600	200

Stoppings with Doors. To reduce the air losses through doors it is necessary:

(a) to build the stopping into a small slot cut into the wall rocks;

(b) to build the doors from two layers of planks with a lining between;

(c) to hang a piece of canvas to the bottom of the door and build a threshold to it;

(d) to make sure that the door fits closely to its frame;

(e) to oppose a large pressure difference, two or even three doors must be used.

If the doors are built in an airway with a rail track passing through it, the floor under the door for about 0.5 to 1 m should be levelled up to the rail head with thick planks, leaving troughs for the wheel flanges to pass In roadways with trolley wires, the doors should be built of two leaves with a small slot between them for the trolley wire, it is also possible to use single-leaf doors, but in this case the top half of the door is cut at the height of the trolley wire (so as to enable the door to be opened) and a small shield is nailed above this opening to close the slot, these doors leak more than two-leaved doors.

This applies also to roads with rope haulage, such as inclines for ventilation, which are also used for lowering timber, a slot for the rope is cut in the bottom of the door; sometimes a threshold is built with slots in it for the rails, to cover the gap between the floor and the door

If a stopping has to be built in a conveyor road, an opening for the conveyor is formed in the stopping, and covered with a canvas apron, to reduce the air losses through this conveyor opening, two stoppings are built, the apron must be hung on the intake side

To eliminate air losses through a water drainage channel under a stopping, a lifting slide gate is set in a pipe drain under the stopping; the gate is lifted just high enough for the pipe to stay full of water; when there is a high pressure drop, the gate must be lowered further so as to avoid the water being blown out of the pipe; lowering the gate creates a small water column opposing the air pressure. Standards for air losses through doors are given in Appendix 5

Air Crossings (see Chapter 16) Air crossings are built for separating two intersecting air flows; crossings built of tubes must be carefully jointed; the supports must be solid and correctly built with a lining of timber flats, in masonry crossing the supports must be carefully fitted to the wall rocks and any vacant spaces must be carefully packed. When the crossing has been built and the air is passing through it, two air measurements should be made in each stream, on each side of the crossing Standards of leakage through air locks are given in Appendix 5.

Partings with Air Locks (see Chapter 16). Partings with air locks are built at the end of a brake incline to reduce the air leakage into it from the main haulage road When the doors are frequently opened during a busy haulage period, or are damaged by bumping with the mine cars, air leakages through these partings may be considerable The walls of these air locks should be built of concrete or brick, with the doors as far as possible from the incline and from each other, so that they are not opened simultaneously; if the air lock is located in a by-pass road, it will leak less than one built in the haulage road.

Surface Buildings and the Collar of the Upcast Shaft. An investigation involving some 30 upcast shafts and the short circuits of air through buildings and ventilating structures at the top of the shaft showed that the losses were very considerable. at main fan installations—20-30 per cent or more of the air coming out of the mine, and at district fans—50 per cent or more Generally speaking, the smaller the equivalent orifice of the mine, the larger, other things being equal, will be the leakages of air.

Measures aimed at reducing short circuits of air through the upcast shaft building include the following

(a) the volume of the building above the upcast shaft must be as small as possible,

(b) the walls of the building must be plastered outside and inside,

(c) the windows must be double-framed (and if the mine head is high, they must be triple), the glass must be carefully placed and puttied,

(d) doors must be double and must satisfy the requirements mentioned above;

(e) if the shaft is used for hoisting, the opening in the wall or the roof for the rope to pass through must be covered with a movable wooden valve fitted to the rope; this valve should be changed when it wears

The collar of an inclined shaft should be lined with brick or concrete, there should be not less than two doors, and for a high head not less than three, old workings, through which air can leak into the incline from the surface, should be carefully blocked or closed off by stoppings. Short circuits can be considered to be "normal" at the following levels.

shafts equipped with main fans, with cage hoisting, 10%,
ditto, with skip hoisting, 15%;
shafts at which auxiliary fans are installed, 20%;
shafts without hoisting, 5%.

Fan drifts Leakages of air at the fan drift occur through the opening for the door, through the slot at the valve for the spare fan, the doors for reversing the air flow (generally with large leakages), and the trapdoors for entering the fan drift. To reduce these leakages into the fan drift it is necessary·

(a) to close the slot for the door in the fan drift at the surface with a small accurately fitted timber panel plastered with clay;

(b) to provide the inspection manholes and any other entries to the fan drift with double, solid trapdoors;

(c) to check that there is no excessive deflection in the fan drift door or valve for the stand-by fan in double installations and to ensure that it does not leave the slots built for it in the fan drift walls; if the leakages are large, they must be stopped;

(d) the doors which reverse the ventilation must be fitted as carefully as possible to their frames; there should be a preference for any method of reversal of the ventilation by which doors are forced on to their frames by the air pressure.

To increase the airtightness of stoppings and other ventilation structures (and also, if necessary, for plastering the walls of airways) V.Ya Baltaitis and Ya L Polesin proposed that they should be plastered with special "jackets" made from mortars containing water glass as setting accelerator, using the following compositions by volume

Description	Water glass	Gypsum	Lime	Cement	Water
Temporary stopping	1.5	2	1	—	2
Permanent stopping	1	—	1	1.5	1 5
Wall rocks in a damp place	0.5	—	1	1	1
Ditto, in a dry place	0.5	1	2 5	1	3

The recommended thickness of the "jacket" is

for temporary stoppings	10-15 mm
for permanent stoppings, depending on their function . . .	15-50 mm
for walls of airways	5-10 mm

The coating is applied by special plant connected to compressed-air cylinders

The mortar can be applied onto steel mesh, hessian sack cloth, etc

Tests of this plaster in laboratory and mine conditions have shown that it is highly efficient.

14-7. CONTINUOUSLY DISTRIBUTED AIR LEAKAGES

This category includes air leakages:
(1) through the worked-out area;
(2) through strip packs;
(3) between roadways driven in pairs;
(4) through the joints of ventilation ducting.

1. Leakages through the worked-out area The extent of these leakages depends on a whole series of factors: the thickness and the angle of dip of the coal seam; the properties of the wall rock; the method of mining; the method of roof control; the depth of the mine. The time factor is also important—under the effect of the rock pressure the waste gradually leaks less and less The joint effect of these factors is complicated and has not yet been sufficiently studied. Let us note merely the following irrefutable statements connected with the working of seam deposits

(1) with reducing thickness and increasing depth of the deposit. the resistance of the waste progressively increases,

(2) the greater the spacing between horizons the greater the resistance;

(3) the more easily the roof breaks, the better it compresses, and the less permeable it is to air;

(4) continuous and careful building of roadside and strip packs increases the resistance;

(5) the construction of solid stoppings and their careful and timely plastering reduce air leakages;

(6) with time, the waste becomes less permeable to air.

The magnitude of the air losses through the waste in metal mines is not yet properly known. Judging from information on air leakages in the mines at Krivoi Rog, and in the Urals, air leakages in these mines are considerable and not less than those in coal mines

The investigation of air leakages through the waste in 65 faces in the Donets Basin (based on the work of the Leningrad Mining Institute) with the faces advancing away from the shafts and the return air moving towards the shafts gave the following standard levels of air leakages:

	Air leakages, per cent of air passing through the face
1 Gently sloping seams (44 faces)	
(a) full caving with strip packs (10-12 m) above the haulage road and below the return airway and with round timber cogs	25-30
(b) partial packing with strip packs (4-6 m) along the roads	15-18
(c) ditto, with advancing direction of the air flow (away from the shaft)	5-6
(d) smooth lowering of the roof	12-15
2 Steep dips (21 faces)	
(a) full caving with a pillar of mineral left above the haulage road and below the return airway	35-40
(b) full packing with pillars left	15
(c) smooth lowering of the roof	10
Round timber cogging reduces the air leakages by about 10%	

Reduction of the air leakages through the waste can be achieved, apart from the methods mentioned above

(a) by full packing;

(b) by retreating working, from the boundary towards the shafts, in these methods, between the haulage road and the return airway

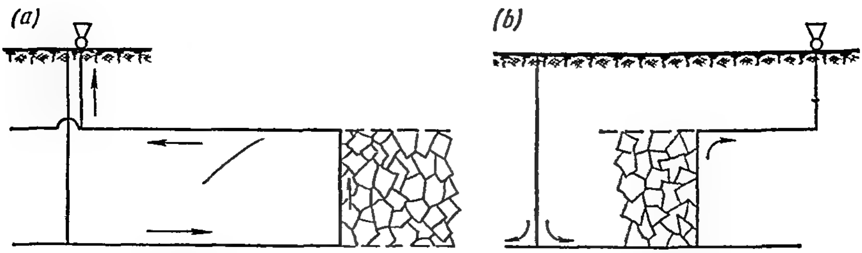

Fig 14-1 Methods of reducing the leakages of air through the waste

there is a pillar of coal through which no air can leak (Fig. 14-1*a*);

(c) by location of the fans at the boundaries of the mine take, the return airway with this arrangement is generally abandoned, and any leakages take place only near the face (Fig 14-1*b*), the total percentage of leakages in the mine is halved;

(d) by increasing the cross section of the two roads so as to reduce the pressure difference between them, since with the quadratic law of air flow the leakages are directly proportional to the square root of the head (and with laminar flow, to the head value), the effect of reducing the head is the most effective reduction of leakages in places where the flow is laminar or near to it And since the tendency of coal left in the waste to spontaneous combustion will increase (within certain limits), in proportion to the air flow

through it, then increasing the dimensions of the roads can be expected to reduce the number of mine fires from spontaneous combustion, and this happens in reality.

Sometimes leakages can be useful, for example, leakages of air through the waste of gassy mines help to ventilate this space and thus lower the quantity of gas accumulated in it; if the gas emission from the wall rocks is considerable in the return airway going back from the face towards the shaft, they reduce the gas concentration in the return airway, similarly, within certain limits, leakages from ventilation ducting, under forced ventilation, to a dead end can also be considered useful because they help to remove the explosive fumes from the drivage.

2 *Air leakages through strip packs* Investigations of air leakages through the waste with various methods of roof control (Leningrad Mining Institute, I I. Medvedev) have shown that this process, for 50-100 m from the face, involves purely turbulent flow; as the distance from the face increases, and consequently with increasing consolidation of the strip packs, the law of air flow at first becomes transitional and then laminar or rather close to it.

Leakage of air through strip packs is also subject to these laws with the difference that because of their higher density by comparison with the broken rock from the roof, leakage can become laminar even at a small distance from the face

The magnitude of the leakages from the face through strip packs depends on the following factors.

(a) the aerodynamic resistance R of the strip packs;

(b) the head h causing the leakage,

(c) the cross-sectional area S of the strip and its width

The head h can be assumed equal to the arithmetic mean of the head at the beginning and the end of the length of the airway in question, the area S is equal to ml, in which m is the seam thickness and l is the length of the roadway along which the leakage takes place. According to A.F. Miletich of the DGI Institute who made detailed investigations of the sizes of stones in the strip packs in the mines of the Donets Basin, the aerodynamic resistance R depends:

(a) on the voids factor in the pack per unit volume, ε m^3/m^3, and on the so-called equivalent diameter of the voids, d_{eq}, through which the air passes

$$d_{eq} = k \frac{\varepsilon}{1-\varepsilon} d_{av}$$

where k = coefficient depending on the shape of the pieces (0 2 to 0.4)

d_{av} = average diameter of the pieces, mm; for various roof rocks, A F. Miletich gives the average values of ε and d (Table 14-2).

TABLE 14-2 Average Values of ε and d_{av} (According to Miletich)

Rock description	ε	d_{eq}, mm	d_{av}, mm	Re for strip packs: loose	Re for strip packs: consolidated	Conditions: Laminar, A	Conditions: Transitional, B	Conditions: Turbulent, A
Very hard limestone	0 45	0 0113	46 4	110	90	0.016	69	0.0030
Hard limestone	0.42	0 0089	41 5	110	90	0 010	128	0 0021
Very hard sandstone	0 41	0 0070	33 8	60	25	0.004	330	0 0006
Hard sandstone and sandy shale	0.38	0 0066	36 5	70	25	0.006	208	0 0014
Very hard clay slate	0 52	0 0143	44.0	130	40	0.014	85	0 0028
Hard carbonaceous and clay shale	0 46	0 0096	38.0	165	80	0.005	242	0 0015
Soft carbonaceous and clay shale	0 37	0.0053	34 0	65	25	0 004	280	0 0006

(b) on the Reynolds number

$$\mathrm{Re} = \frac{d_{eq} v}{\nu}$$

in which v = actual air velocity in the free cross section

$$v = \frac{nQ}{\varepsilon m l} \text{ m/sec}$$

where Q = air flow through the face, m³/sec
n = coefficient of air loss equal to

$$n = \frac{Q_{road} - Q_{face}}{Q_{face}}$$

In columns 5 and 6 of Table 14-2 the values given are the critical Reynolds numbers up to which the flow remains laminar in loose (column 5) or consolidated (column 6) packs

Considering the process of air leakage A F. Miletich, using the results of air tests through various layers, proposes the following equations for calculating the width b of strip packs.

(a) in the laminar region and the non-steady conditions:

$$b = \frac{Ah}{v} \text{ metres and } b = \frac{h}{Bv} \qquad (14\text{-}5) \text{ and } (14\text{-}6)$$

(b) in the turbulent conditions:

$$b = \frac{Ah}{v^2} \text{ metres} \qquad (14\text{-}7)$$

where A and B are coefficients the values of which are given in Table 14-2.

Example. Calculate the width of a loose strip pack on the following basis. face length $L = 150$ m; seam thickness $t = 1.2$ m, the cross section of the upper and lower roads is 5 5 m², the cross section of the face is 3 6 m², the coefficient of resistance of the roads is 0.0017 and of the face 0.0040, the leakages of air in a length $l = 100$ m should not exceed 10%; the quantity of air entering the district is 12 m³/sec; the roof rock is a strong limestone

Solution. Let us determine the pressure drop across the strip pack at the beginning and the end of the 100-m length of the roadway, the head in the roadway is equal to

$$h_{road} = 2\alpha \frac{LP}{S^3} Q_{road} Q_{face} = 2 \times 0.0017 \frac{100 \times 9.75}{5\ 5^3} \times 12 \times 10\ 9^* = 2\ 6 \text{ mm}$$

the head in the face.

$$h_{face} = 0.0040 \times \frac{150 \times 8.4}{3.6^3} \times 10.9^2 = 12.9 \text{ mm}$$

Consequently the pressure drop across the strip pack at the beginning of the length will be equal to $12\ 9 + 2\ 6 = 15\ 5$ mm and in the face 2 6 mm, the average being 9 05 mm**

Let us determine the actual air velocity through the pack and its corresponding Reynolds number.

$$v = \frac{nQ_{face}}{\varepsilon ml} = \frac{0.1 \times 10.9}{0.45 \times 1.2 \times 100} = 0\ 02$$

$$\text{Re} = \frac{d_{eq}\ v}{\nu} = \frac{0.0113 \times 0\ 02}{14.4 \times 10^{-6}} = 15\ 7$$

where n equals 0 1 (10%), ε equals 0 45 (from Table 14-2 according to A F. Miletich), $d_{eq} = 0\ 0113$; since $\text{Re}_{crit} = 110$ (with loose filling, Table 14-2), then laminar flow takes place, thus, according to Equation (14-6) we obtain the width of the strip pack b

$$b = \frac{Ah}{v} = \frac{0.016 \times 9\ 05}{0.02} = 7.2 \text{ m}$$

to which we must add 0 5 metre for the large lumps of stone each side of the wall, in all, $7\ 2 + 2 \times 0\ 5 = 8\ 2$ m.

$$^* \ Q_{face} = \frac{Q_{road}}{1+n} = \frac{12}{1+0.1} = 10\ 9$$

** In reality, since the curve of change of pressure drop along the roadway is concave, the theoretical average head will be below the arithmetic mean.

Since the data in Table 14-2 may not correspond to the actual values, as a basis for calculations they should be multiplied by a safety factor of about 1 2 to 1 3

Using the data from the formulas, we can solve the reverse problem, that is, given a certain width of strip pack, to determine the percentage of air lost through it

3 Air losses between roadways driven in pairs This concerns headings driven to the strike in the form of a pair of roads, as well as rises or dips driven with manways parallel to them When these are long, the leakages between them from the intake to the return road can be considerable The amount of the leakages depends on the spacing between the stoppings, on their perimeter, on their permeability to air, on the composition of the surrounding rock, and on the pressure drop between the two roads

These leakages were very thoroughly investigated by M A. Patrushev of the Leningrad Mining Institute in a number of mines of the Donets Basin To calculate the air quantity lost, he developed the following equation

$$Q=Q_k+0\ 5AB+\sqrt{ABQ_k+Ah_e} \qquad (14\text{-}8)$$

where

$$A=\left(\frac{m\sqrt[3]{L}}{\psi\sqrt{R_{th}}}\right)^2 \text{ and } B=0\ 5\,(r_1+r_2)\,LQ_k$$

In these equations L = length of the paired roads, m

r_1 and r_2 = resistances per metre length of these roads, kilomurgs

m = number of stoppings

$$\psi=\frac{9\ 3}{P}$$

TABLE 14-3 **Values of R_{th} for Solid and Fissured Rocks**

Type of stopping	Values of $R_{th}\times10^{-3}$	
	solid rock	fissured rock
Slag and rubble concrete	390	120
Masonry	290	90
Slag-block and brickwork	220	70
Round timber	120	40
Planks	70	22
Round timber + rubble wall	40	10

in which P = perimeter of the stoppings, m
Q_k = quantity of air flow
h_e = pressure drop at the end of the roadway
R_{th} = theoretical resistance of the stoppings separating the two roadways, per metre length; the values of R_{th} for solid rock and fissured rock are given in Table 14-3.

Example. Given· the length of the roadway 1,000 m, the spacing between stoppings 50 m; the final quantity of air 20 m³/sec and the head 30 mm, the perimeter of the stoppings 10 6 m, the specific resistance of the two parallel roads 2 5 × 10^{-5}, the stoppings are slag concrete in fissured rock. Find the air leakages

Solution. Let us determine the values of A, B and AB

$$A=\left(\frac{20\sqrt[3]{1{,}000}}{0\ 877\sqrt{120{,}000}}\right)^2=0\ 436$$

and

$$B=0\ 5\times 2\ 5\times 10^{-5}\times 1{,}000\times 20=2\ 5$$

$$AB=0\ 436\times 2\ 5=1\ 088$$

Substituting these values for A and B in Equation (14-8) we obtain:

$$Q=20+0\ 5\times 1\ 088+\sqrt{1\ 088\times 20+0\ 436\times 30}=20\ 45\ \text{m}^3/\text{sec}$$

Consequently, the air leakages are equal to 26 45—20=6 45 m³/sec or as a percentage

$$p=\frac{6\ 45}{20}\times 100=32\ 2\%$$

An approximate method of calculating the air losses in paired roadways consists in multiplying the number of stoppings by the percentage of air lost through each stopping (in relation to Q_k); these percentages are given below

Type of stopping	*Solid rock*	*Fissured rock*
Slag and rubble concrete	0 80	1 45
Slag block .	0 85	1 50
Round timber	1 15	1 95
Sawn timber	1 40	2 50

Applied to the example above,

$$p=20\times 1.45=29\ 0\%\ \text{instead of}\ 32.2\%$$

14-8. AIR LOSSES WITH UNDERGROUND BOOSTER FANS

Underground booster fans create an increased pressure downstream, while upstream, compared with the original pressure, there is a reduced pressure. This can often create extremely undesirable recirculation of the air flow at the fan.

Let the roadway AC (Fig. 14-2a) be the intake to the face and the roadway DE be the return from the face; let there be a booster fan at the point B and, as a first approximation, let the head vary as a straight line; then plotting below the straight line OO the value of the head, and above it the value of the positive head along both roads, we obtain a diagram (Fig. 14-2b). This diagram shows that at the length AB, the air will be sucked out of the return airway into the intake. The effect of the main fan is indicated by

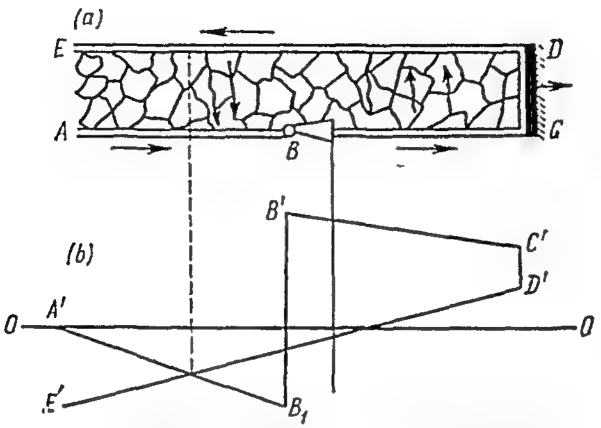

Fig 14-2 Analysis of the air leakages caused by an underground booster fan

the straight line $D'E'$ intersecting the straight line $A'B'$ in the interval between the points A and B (Fig. 14-2a) where the suction effect ceases.

The diagram (Fig. 14-2) enables the following conclusions to be drawn:

1) The nearer the booster fan is to the entrance to the roadway, the smaller will be the air quantity sucked from the return airway down to the haulage road.

2) The nearer the point of installation of the fan to the face, the greater will be the probable length along which air will be sucked from the return airway down to the intake; but since near the face the resistance of the waste is considerably less, the volume of air pushed out of the lower road into the upper one will be larger than the volume of air sucked down from the upper road.

3) To completely eliminate recirculation, a booster fan should be installed in the return airway where the broken rock in the waste is fully consolidated. This is possible in non-gassy mines and metal mines, in gassy mines the installation of a fan in the return airway, although undesirable, is allowed when the equipment is explosion-proof.

14-9. EFFECT OF AIR LEAKAGES ON THE WORK OF THE FAN. SHORT CIRCUITS

Let us first consider *short circuits*; this implies the short-circuiting of all or part of the ventilating air stream, avoiding the main ventilating circuit of the mine; for example, when doors are opened at the pit bottom, connecting the downcast shaft with the upcast shaft, a short circuit occurs between the two pit bottoms; when the doors over the top of the upcast shaft are opened, a short circuit occurs directly between the fan and the atmosphere.

Short circuits are extremely undesirable in mine ventilation; their main effect is a reduction of the air flow through the ventilating circuit rapidly with all the ensuing consequences; secondly, the

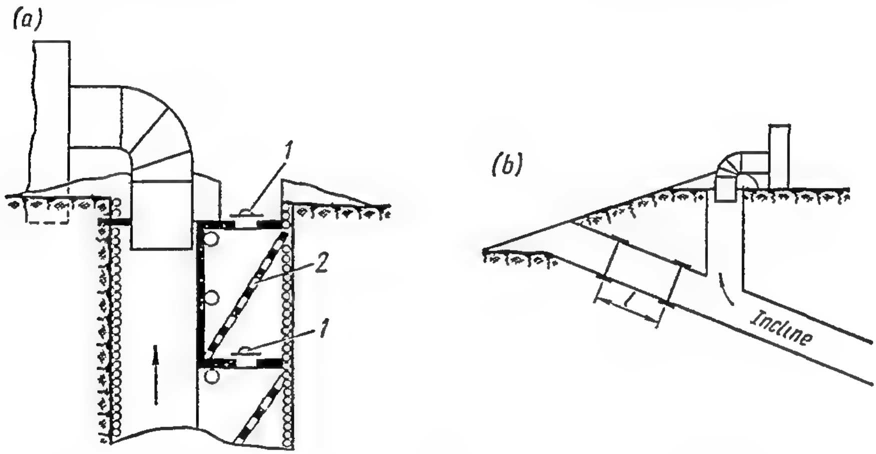

Fig 14-3 Methods of connection of district (surface) fans
(a) to a small pit, (b) to an incline, 1—steel trapdoors for men, 2—ladders

fan output greatly increases and consequently the motor can overheat (with axial-flow fans working on the right half of the characteristic this danger does not exist). When the two doors in an inclined shaft are opened to let a train of mine cars pass, the motor does not generally overheat because it is usually powerful enough, but in addition, for small, and sometimes even medium-size fans the fan drift has adequate resistance; even in the absence of the whole mine circuit, the fan drift resistance is enough to prevent the fan working against zero resistance.

Figure 14-3*a* and *b* shows recommended methods of connecting the auxiliary fans at small pits and inclines, ensuring adequate airtightness of the connection and preventing short-circuiting when timber is being lowered. If the trapdoor is carefully built, with the

doors and frames on which it bears, the air leakages should not exceed 15-20% of the air flow passing from the mine.

Short circuits can be prevented by strictly observing the ventilation regulations; in addition, in particularly important places interlocks should be built in, to prevent the opening of a second door at the same time as the first one

The effect of leakages on the work of the fan The presence of leakages underground shows that in addition to the functional paths of the air, it can find other paths The total resistance against which the fan works will be diminished by the presence of these leakages, but the leakages as they increase will also increase the fan output; these increases will only slightly be compensated by the reduction of the air flow to the face Conversely, with a reduction of leakages insofar as the fan output will fall, it is possible usually to count on only a small increase of the air flow reaching the face. On the other hand, the power consumption can often be considerably reduced by reducing the air flow passing through a radial-flow fan.

14-10. CALCULATION OF THE TOTAL LEAKAGES IN THE MINE

The values of absolute leakage through ventilating structures indicated in this chapter, including stoppings in paired roadways, apply to structures in good condition from the viewpoint of impermeability to air Therefore, they can be used for calculating the expected air losses in the mine as a whole. It is necessary to calculate the number of these ventilating structures (stoppings, air crossings, etc) and, using the data given above for standard leakages in m^3/min through one structure, to determine the total leakages of air through the mine as a whole To these must be added the losses through the waste and through the strip packs

Calculation of the air losses in this way for some 30 working mines in the Donets Basin has shown that safety factors of about 1 4 to 1 6 are fully acceptable (safety factors for air are discussed in Part Three, page 527); in other words, the losses throughout the mine constitute 40 to 60% of the air flow reaching the faces, provided that the losses through the ventilating structures do not exceed the standards indicated in this chapter.

CHAPTER 15

VENTILATION OF DEVELOPMENT DRIVAGES (AUXILIARY VENTILATION)

15-1. GENERAL

This chapter is devoted to methods of ventilating development roads and dead ends and the equipment used for this purpose—a subject described as *auxiliary ventilation*

The problem of ventilating drivages or dead ends consists in delivering along the roadway a comparatively small quantity of air at a distance of some tens or hundreds or thousands of metres, and removing it usually by the same roadway

The equipment for supplying or removing the air (generally ventilation ducting) must be constantly lengthened and brought nearer to the face so as to sweep it by a positive air flow.

Auxiliary ventilation of mine roadways can be achieved by:

(a) the head developed by the main mine fan or the natural draught,

(b) by so-called auxiliary fans,

(c) by the use of injectors,

(d) by a combination of methods (b) and (c)

The methods used for supplying the air to the face and bringing it away include

(a) ventilation by line brattices,

(b) paired parallel roads separated by a mineral pillar;

(c) parallel roads formed in driving along a wide face, with a pack built between them,

(d) ventilation ducting.

(e) drilled holes

In addition, particularly at metal mines the face is often ventilated with compressed air.

According to the Safety Regulations now in force, "the ventilation of the faces of development roads must be effected by means of the mine head or by auxiliary fans", but auxiliary fans are allowed only in "special projects subject to the fan operating continuously".

The ventilation of drivages by means of the main mine fan head is much more reliable than using auxiliary fans The main fan head is most often used when the additional pressure difference

h_{add} needed for the ventilation is not great; in the opposite case, on the basis that the additional power spent is equal to

$$\frac{h_{add}Q}{102\eta}$$

in which Q is the total quantity of air passing through the fan, it is considerably more favourable to use a small auxiliary fan in which the power consumption will equal

$$\frac{h_{add}q}{102\eta}$$

where q is the output of the small fan.

The ventilation of roadways in gassy mines by diffusion alone is forbidden by the Soviet Safety Regulations, and in non-gassy mines is authorized for a distance of only 10 metres Depending on the layout of the dead end in relation to the intake, Professor A.M. Karpov distinguishes between (a) straight; (b) side, and (c) reversed dead ends, for example, in the return airway.

Each type of dead end has different steady eddying zones with vigorous gas exchange as well as a zone of "frequent eddies". The lengths of the zones of stable eddying, according to the experiments of A M Karpov on the three types of dead end listed above, are 3-5b, 3-4b, and 2-3b, in which b is the width of the dead end. The lengths of eddying zones can be somewhat increased by placing baffles in the intake airway at an angle to it and directing the air into the dead end

15-2. VENTILATION BY A LINE BRATTICE

Canvas brattices consist of a row of timber props with canvas nailed to them To reduce air leakage, the canvas is nailed with broad-headed nails, to a batten fixed to the roof; at the bottom it is weighted to the floor by small coal or rock dust. When the separate lengths of canvas are put up, they must overlap each other. The advantage of these brattices is that they are quick and cheap to build; their disadvantage is large air loss, reaching 80 per cent in 40 metres.

Timber brattices are built from half-round timbers or boards nailed to props from the top down (Fig. 15-1a); the joints are filled with clay. Sometimes the boards are butted, and the joints between them are covered by nailing battens over them (Fig. 15-1b); for greater airtightness, boards are nailed on both sides, and the space between them is rammed full of sand, clay, etc.

The greatest airtightness is obtained by using clay, stone dust and fine sand Guniting is another way of making brattices airtight.

In roadways subject to high rock pressure, there should be a plastic clay cushion at the roof. Figure 15-1*c* shows a line brattice in a steep seam.

The advantages of timber brattices are their low cost, high speed of erection, and ease of repair; their disadvantages include rapid falling into disrepair and the leakage of a large amount of air flow.

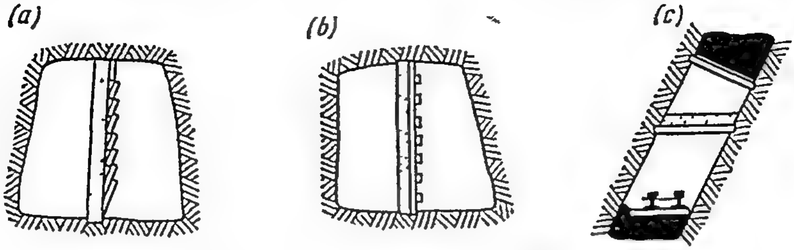

Fig. 15-1. Timber line brattices

Masonry brattices, for example brick or concrete, are nowadays rare; to obtain the required resilience they are packed with timber; and to protect them from damage by shotfiring, vents with flaps are built into them.

The general disadvantages of line brattices are that they hinder movement along the roadway and air losses are high; through brickwork they reach 10 to 20 per cent every 100 m.

15-3. VENTILATION BY PAIRED ROADWAYS

This method consists in driving parallel to the main road, at a distance of 10 to 20 metres (in weak rock or coal at depths below 300-400 m up to 30 to 35 m), another road as a return airway (Fig. 15-2). At a definite spacing (according to the Safety Regulations not more than 30 m) the roads are joined by stentons across which temporary timber stoppings are first built, and later changed for heavy round timber or masonry stoppings. Short dead ends are ventilated by line brattice beyond the stentons as shown on the drawing; if the extension is long, auxiliary fans are used

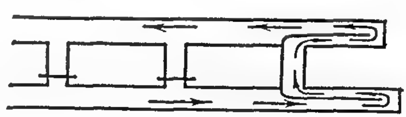

Fig. 15-2 Diagram of ventilation by paired drivages

This method of ventilation is widely used in seam deposits but rarely in metal mines; parallel roadways are joined in metal mines by short cross-measure drifts.

The advantages of this method include simplicity, the possibility of giving the face a comparatively large amount of air at a small head, and the fact that the roadway is not blocked throughout

its length Its disadvantages include large losses of coal in the pillars; the inconvenience of coal delivery to the haulage road from the lower sub-level; the high cost by comparison with driving a single road The efficiency of the ventilation depends on the amount of the air losses through the stoppings (see Chapter 14)

When the mineral pillars are not large enough, they crack and split, increasing the air losses through them which may lead to spontaneous fires in the coal.

To reduce coal losses in the pillar the method is sometimes changed for the use of strip packs which naturally increases the air leakages Alternatively, instead of stentons, the two parallel roads may be joined by large boreholes; if the holes are long (for example, if a haulage road and a return airway are being driven in a steeply dipping seam in the Kuznets Basin) auger drills are used, of СБМ-3у type, and if the holes are short, light ЛБС-2 rock drills are used which are described in the literature. The hole diameters are from 150 to 250 mm, reaching 850 mm after reaming. For drilling holes from above downwards the БШ drill rig designed by Dongiprouglemash is used.

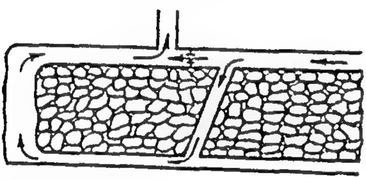

Fig 15-3. Diagram of ventilation by a road through a pack

15-4. VENTILATION BY ROAD THROUGH THE PACK

This method in driving roads on a wide face with a pack between, consists in the following part of the intake air is directed by a stopping with a door or by a simple brattice sheet into a so-called gob road (Fig 15-3) for ventilating the face; sometimes the gob road contains a small fan working through a stopping.

The disadvantages of the method include large air losses; they can be reduced by careful facing of the roadside packs with clay, or by lining them with timber slabs

15-5. VENTILATION DUCTING

Underground roadways nowadays are ventilated by means of ducts made of the following materials timber, metal, textile, and various synthetic materials

1. Timber Ducting. Timber ducting is made from boards or plywood The first type is rectangular, being built of boards nailed together and connected by couplings, made airtight with clay; the coefficient of friction, counting the joints and other factors, should be assumed not less than 0 0004 to 0.0005 Timber ducts pass a large

quantity of air and are used sometimes when other types are not available

Plywood ducting is often circular and is lighter than ducting made from boards, the joints are also made by couplings It passes large quantities of air but less than board ducting; it resists acid water better than metal ducting, and its coefficient of friction is 0.0003 to 0 0004 or more. It is used in salt mines and copper pyrite mines where steel tubes would rapidly become useless.

No standards of air leakage for timber ducting can be given; it can reach 30 to 50 per cent or more per 100 m.

2. **Metal Ducting.** Metal tubes are made of sheet iron or steel, from 1 to 3 mm thick, depending on the diameter The sheets are usually welded, rarely riveted; the ducting diameter is 200 to 800 mm, occasionally more; the length of one piece of ducting is usually 2-3 m (for shafts, up to 4 m)

The coefficient of friction of steel ducting underground can be taken to be from 0 0003 to 0.0005, depending on the diameter For calculations it is more convenient to use the values of resistance in kilomurgs which are given in Table 15-1 for ventilation ducting lengths of 100 m

TABLE 15-1 **Resistance of Steel Ventilation Ducting (kilomurgs per 100 m)**

Air velocity, m/sec	Diameter of ducting, mm				
	300	400	500	600	800
5	105-125	23-28	7 3-8.8	2 8-3 3	0.6-0.7
10	97-117	21 4-25 6	6 7-8 1	2 5-3 0	0 56-0 67

The smaller values apply to straight lengths built of clean tubes without dents; the larger values apply to lengths built of old battered ducting, if the coefficient of friction of smooth straight new ducting is 0.0002, then poor alignment increases the resistance by 10 to 50%, projecting washers increase it by 50 to 70%. dents by up to 100%, rust by 30 to 80% (with a rust thickness of 2 to 3 mm). sometimes the actual diameter of the ducting does not correspond to the nominal diameter, which results in an abrupt change in the resistance of the ducting.

The aerodynamic resistance of ducting is affected by the following factors

(a) under forced ventilation, the resistance is somewhat larger than for exhaust ventilation because of the high turbulence of the air flow;

(b) for an exhaust installation, leakages into the ducting increase the resistance because of the disturbance to the air flow caused by the small jets of air leaking in

Methods of Joining Ventilation Ducts Underground, various methods of joining steel ducting are used, the commonest being: (1) a joint resembling the spigot and socket joint (like drain pipes) with one end fitting into the other; (2) a flanged joint. Other joints also exist, for example, those using rubber rings.

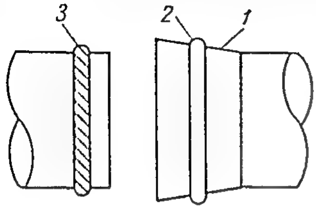

Fig 15-4 Joint for ventilation ducting, made by a conical end-piece

The *spigot and socket joint* is simple to use but inconvenient when dismantling the ducting for repair; it has large air leakages, the joints are made tight with clay or cement mortar, a simple but effective joint is shown in Fig 15-4, onto one end of the ducting a small conical connection *1* is welded with an internal ring-shaped recess *2* in which, when the pipes are joined, the rubber ring *3* of 16 mm diameter is pushed, from another ring-shaped groove on the end of the other pipe, the joint takes 3-4 min to make, there was only 6-7% air leakage in one length of ducting 375 m long consisting of 100 separate lengths

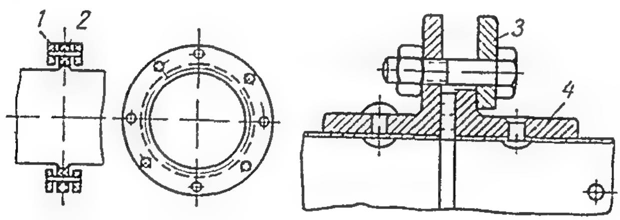

Fig 15-5 Joint for ventilation ducting with one loose flange

1—flange, *2*—rubber washer, *3*—loose flange, *4*—steel angle section

The ordinary *flanged joint* is shown in Fig 15-5, for convenience of assembly one flange is a loose ring bearing on a fixed flange on the end of the ducting; the packing used is cardboard soaked in grease, or a rubber ring or simply a piece of hemp rope, the flanges are joined by 6 to 8 bolts, this type of joint is better than the previous, but in underground conditions the air losses are nevertheless high and with careless assembly they reach 20 to 30% per 100 m The reason for these high leakages is that the flanges deform comparatively quickly and the thin packing (3-5 mm) does not compensate for the denting of the flanges

Much better flanged joints are shown in Fig. 15-6*a*, *b*, and *c*. In Fig. 15-6*a* between the flanges is a flat metal ring *1*, on the inner side of which a rubber bead 16 to 25 mm thick is fixed, fitting the

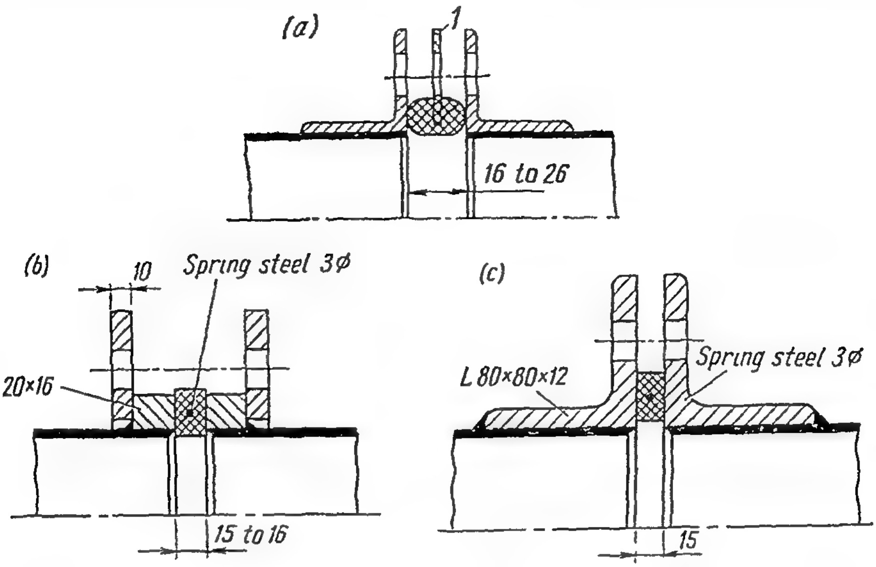

Fig 15-6 Various types of flanged joint for ducting

flanges at the place where they are least liable to deform; the ring has 3 to 4 bolt holes; in the second type of joint (Fig. 15-6*b*) the rubber packing 15 to 16 mm thick is pressed between two metal rings; a ring of 3 mm diameter spring steel gives the necessary stiffness to the packing; the joint of Fig 15-6*c* differs from that in Fig. 15-6*b* by the absence of a steel ring. All these joints are highly airtight even with only 3 to 4 bolts.

Metal ducting on assembly is hung to the supports by wire or on timber brackets; occasionally it is placed on trestles.

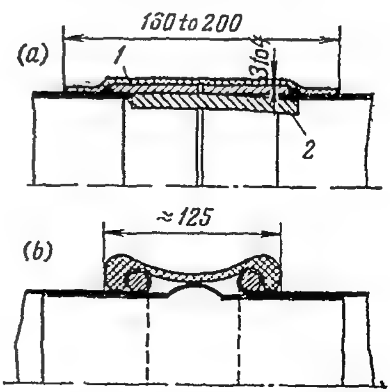

Fig 15-7 Ring joint for ducting: *1*—sleeve, *2*—steel dowel rod

Two types of joint using *rubber rings* are shown in Fig 15-7*a*, and *b*. The first one is fairly widely used in the Belgian coal mines; on the ends of the ventilation tubes stiffened by metal rings welded to them, there is a rubber collar fixed on the tube, the joint is strengthened by 3-4 dowel rods welded to one tube and passing on assembly into the other tube.

The second type of joint is used in Germany in the Saar; for this type of joint use is made of a metal connecting sleeve entering both ends of the ducts to be joined, a metal ring of thick wire placed on the bent-back ends of the ducting, and of a shaped rubber sleeve with thickened edges

3. **Textile Ducting.** Ventilation ducting made from several types of textile, including rubberized types, is made in lengths of 5-10 up to 20-30 metres and with diameters from 300 to 600 mm. It is extremely light as compared with metal ducting, weighing only 1 kg per metre length of ducting 300 mm in diameter, is rapidly installed, easily adapted to bends in the roadway, and leaks less than metal ducting. Its disadvantages include low strength, and it is thus easily damaged by puncturing or slitting; in addition, it cannot be used for exhaust ventilation systems (unless it has strong wire stiffening rings) Finally, its airtightness depends in a large degree on the air pressure; if the pressure is small, the joints are not very airtight and they leak; similarly at low pressure the resistance rises rapidly when the ducting is hung up, forming folds

Nevertheless, this ducting is widely used nowadays and subject to adequate internal pressure and absence of damage it works well with less leakage than metal tubes.

4. **Ducting Made of Rubberized Textile. Type M Ducting.** This ducting is nowadays widely used in Soviet mines. The textile is rubberized on both sides and is 0 8 to 1.2 mm thick. Its breaking strength is not less than 25 kg/cm^2 which allows it to be used for the following maximum pressures at a diameter of 500 mm—700-750 mm water; at a diameter of 600 mm—500-600 mm water (obtained by several fans operating in series).

Lengths of this ducting are joined together extremely simply (Fig 15-8) using spring rings *9*, in the ends of the ducting; one end of the ducting is squeezed by the hands into the end of the other length of ducting, after which the rings extend and are clamped together with a loop *5*. The joint is covered by a hinged metal hoop *1*, with a locking lever *3* In vertical shafts the ducting is hung on loops *4*, in horizontal roadways it is hung by the hooks *7* from a taut rope.

Because of the small number of joints, the air leakages from this ducting are less than those from metal ducting with ordinary jointing

The ducting is hung from hooks to a taut rope 5-6 mm in diameter. A metal rack with holes through it serves to hang the ducting with hooks passing through the holes So that the ducting shall not sag the rope should be supported every 4-5 m at least.

5. **The PVC-Coated Cotton Ducting Developed by VNIIOMShS.** The material (tent duck) is coated inside with PVC, and outside

with chlorinated PVC lacquer. Its coefficient of friction is 0.0002. One of the methods of jointing this ducting is shown in Fig. 15-9; the joint is made with a metal coupling *1*, to the ends of which are welded hoops of round bar 15 mm in diameter. To this coupling the ends of the ducting are fixed in such a way that one length covers the next by 10 cm, the ducting then being clamped to the coupling with a metal hoop *2*. This joint is airtight; the losses in 100 m are about 7 per cent.

If the ducting is torn, it can be patched with a solution of chlorinated PVC resin and dichloroethane.

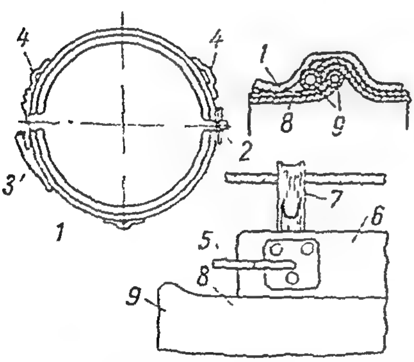

Fig 15-8 Jointing of rubberized textile ducting by spring steel rings.

1—steel clamp, *2*—hinge, *3*—adjustable lever locking device, *4*—suspension loops, *5*—tensioning loop, *6*—suspension plate, *7*—suspension hook, *8*—ducting, *9*—steel ring

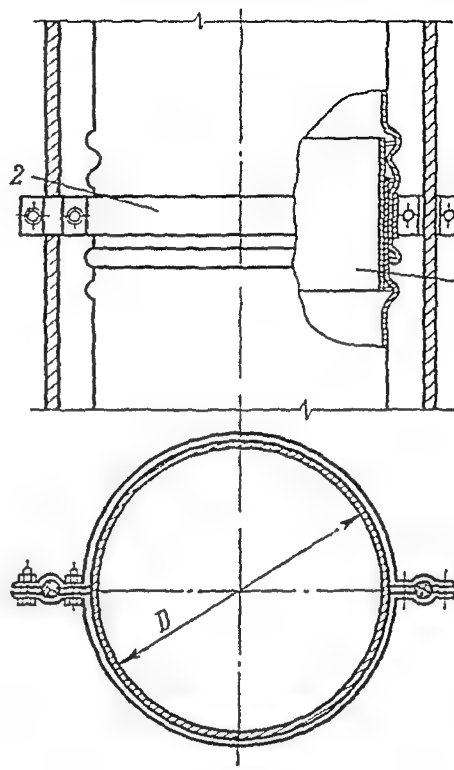

Fig. 15-9. PVC-coated cotton ventilation ducting

Table 15-2 shows the resistance of type M ventilation ducting of various diameters and lengths.

Air Leakages. The degree of airtightness of mine ventilation ducting is the deciding factor in its use; in short lengths losses reaching 50 per cent of the fan output are tolerated, but in long lengths of several hundred metres, extremely large losses are a serious hindrance to ventilation, and in the case of particularly long ones (several kilometres) they are a threat to the use of ducting for ventilation. The reduction of air losses in ventilation ducting therefore needs the most serious attention.

The law of air leakage through mine ventilation ducting varies from purely quadratic with fairly large leakages, down to nearly laminar with the joints made of rubber sleeves, carefully clamped with metal rings; in other words, the power changes with the amount of air leakage from 2 to 1 If the joints are made in such a way that increase of pressure opens the leaks further, for example, with rubber rings without metal hoops, or if the ducting material leaks more (textile ducting), the power can be even less than unity.

TABLE 15-2 Resistance of Type M Ventilation Ducting (kilomurgs)

Ducting length, metres	Diameter of ducting, mm		
	400	500	600
100	30	10	4
200	56	18	7
300	81	26	10
400	102	33	13
500	123	40	15 5
700	161	53	20
1,000		64	24 4
1,200		72	27 4
1,500		76	28 8

The absolute value of air leakage through the joints depends on the following factors

(a) the tightness of the joint, i e. its type and the care with which it has been made;

(b) the diameter D of the ducting; in other words, on the perimeter through which the leakages occur,

(c) the length of one unit of ducting l;

(d) the pressure inside the ducting, which depends on (a), (b), the length L and the air flow Q.

The air leakages can be calculated from the formula of VNIIOMShS

$$\frac{Q_f}{Q_0} = \left(\frac{kD}{3}\frac{L}{l}\sqrt{R}+1\right)^2 \qquad (15\text{-}1)$$

where Q_f and Q_0 = air flows in the fan and at the end of the length of ducting

D = ducting diameter, m

l = length of one unit of ducting, m

L = overall length of ducting, m

R = aerodynamic resistance of the whole length, kilomurgs

k = coefficient of permeability of the joints, equal to the quantity of air in m³/sec passing through a joint of arbitrary ducting 1 m in diameter under a pressure difference of 1 mm water (also known as the leakage factor)

Table 15-3 gives values of the coefficient k for the permeability of the joints for the condition when

(a) the percentage air leakage is known per 100 m length of ducting;

TABLE 15-3 **Values of the Leakage Factor at Joints, k, (Coefficient of Permeability), in Relation to the Percentage Losses per 100 m Length**

Per cent leakage	$k\ 10^3$	Per cent leakage	$k\ 10^3$
1	0 222	15	3 333
2	0 444	20	4 444
3	0 666	25	5 555
5	1 111	30	6 666
7	1 555	40	8 888
10	2 222	50	11 110

In general, $k\ 10^3 = 0\ 222\ p$

(b) the joint type is known.

The actual values of the permeability of joints, found from tests for air leakages underground, are shown in Table 15-4

TABLE 15-4 **Actual Leakage Factor, k**

Description	$k\ 10^3$
Metal ducting for ventilation	
1 Spigot and socket joints, made tight with clay	7 43
2 Ditto, made tight with cement	3 49
3. Flanged joints	
(a) joint quality "ordinary underground"	5 0
(b) rubber washers, with good joints, and bolts tight	2 2-3 0
(c) ditto, very good joint, very carefully tightened	1 0
(d) ditto, rubber washers of the type shown in Fig 15-6	0 034
Textile ventilation ducting	
1 Rubberized textile, type M . . .	1.57
2 Cotton tent duck, coated with PVC, VNIIOMShS type	1 57

Table 15-5 shows calculated values of the ratio $m = \frac{Q_f}{Q_0}$ for type M ducting, bearing in mind the fact that with an increase in length of ducting and consequently with its pressure, the air leakages diminish

TABLE 15-5 Values of the Ratio $m=\frac{Q_f}{Q_0}$ for Type M Ducting

L, metres	100	200	300	400	500	700	1 000	1,200	1,500	2,000
m	1 075	1 137	1 19	1 25	1 30	1 39	1 54	1 76	2 085	2 63

Type M rubberized textile ducting has been investigated by N Bogomolov; his observations show that

(a) with increasing pressure the resistance of the ducting diminishes (because the ducting is straightened), but from 300-400 mm the value of the coefficient of resistance remains constant; at very large pressures (1,000 mm or more) the ducting inflates like a string of beads and its resistance rapidly increases;

(b) the ducting material practically has no leakage (at pressures up to 600 mm); leakages through the joints are subject to laminar flow, they vary with the length of the line of ducting.

The calculations on rubberized textile are performed in the following sequence.

(1) Air leakages are determined from the formula

$$\Delta Q = k_1 L^{1\ 85} \tag{15-2}$$

where k_1 is a coefficient depending on the air quantity flowing from the ducting outlet at the face, its value is determined from Fig 15-10

The fan output required is

$$Q_f = Q_{face} + \Delta Q$$

(2) The pressure which the fan must create is calculated from the equation

$$h = kQ_f^{1\ 55} \tag{15-3}$$

where k is a factor depending on the length of the ducting, its value is obtained from the graph in Fig 15-11

When the ducting passes round bends, its length must be increased in the calculation; for a 45° bend it is increased by 10 diameters and for a 90° bend by 20 diameters (the so-called equivalent length of the bend).

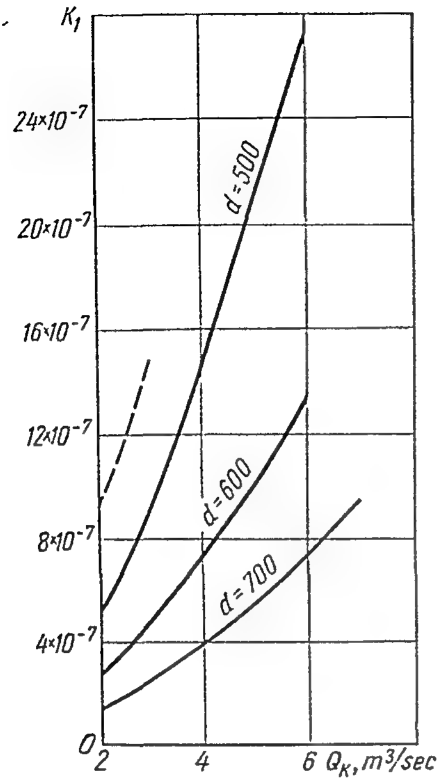

Fig 15-10 Curves for the coefficient k_1 in Equation (15-2)

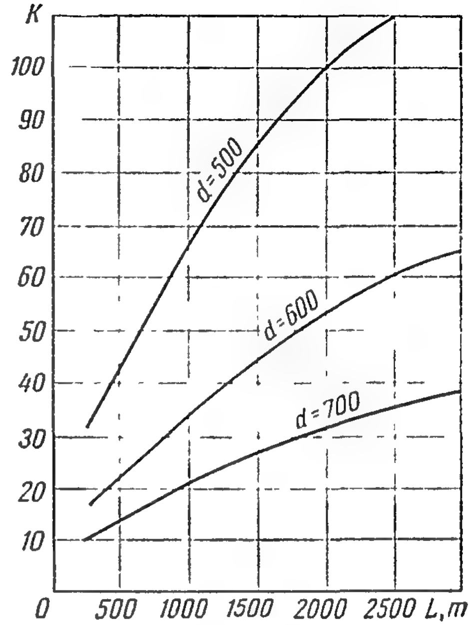

Fig 15-11 Curves for the coefficient k in Equation (15-3)

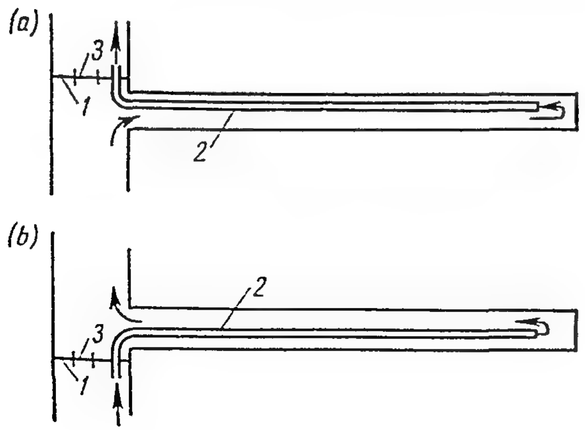

Fig 15-12 Methods of ventilating a dead end by ventilation ducting

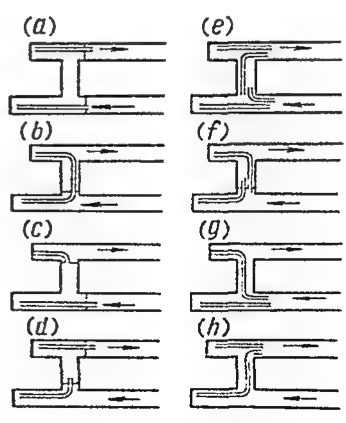

Fig 15-13 Methods of ventilating paired drivages using ducting

Various methods of using ducting for ventilation Figure 15-12*a* and *b* shows how a dead end is ventilated from an intake airway from which the dead end branches off· *1* indicates a stopping, *2* is the ducting, *3* is a door with a regulator, the regulator size can be varied to vary the air flow along the ducting

In the ventilation layout used in Fig 15-12*a* the dead end is ventilated by intake air, but near the face, a stagnant volume of

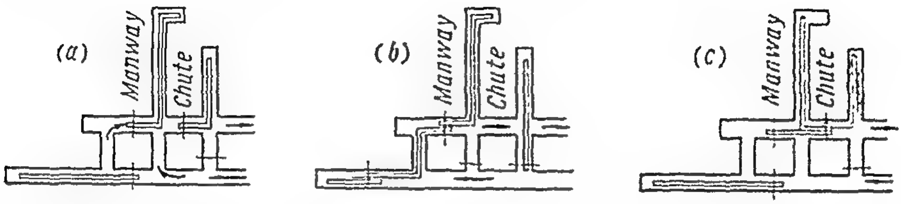

Fig 15-14 Layouts for ventilating the faces of a horizontal drivage, a chute and the manway parallel to it, without using an auxiliary fan

air remains from which the gases formed in shotfiring or emitted from the face are only slowly pulled out, in the layout of Fig. 15-12*b* the air passes out of the ducting at a certain velocity and with a positive flow into the face, forcibly sweeping the gas from the dead end; the stale air, however, moves back along the drivage and not through the ducting In practice, both layouts are used, but the second one is the more usual, in particular for ventilating after shotfiring because it enables the working place to be more quickly cleared of poisonous blasting fumes

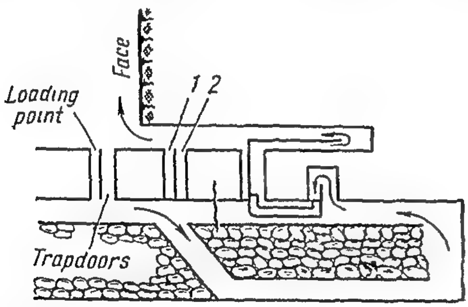

Fig 15-15 Method of ventilating the dead ends of a horizontal roadway in the seam and a chute and manway
1—manway compartment, *2*—coal compartment

Various ventilation layouts of paired roadways using ducting are shown in Fig. 15-13 series layouts for the faces (*a*, *b*, *c*, and *d*), and splits (*e*, *f*, *g*, and *h*) Especially desirable are those layouts which have no stopping in the haulage road or the road parallel to it

Figure 15-14*a* shows ventilation by ducting, without an auxiliary fan, to the faces of a haulage road, a manway, and a chute, these are all ventilated in series, although the air passing out of the manway is diluted by some intake air coming through the stenton along the haulage road Fig 15-14*b* shows splits for ventilating all three

faces but with the advantage of Fig 15-14*a* maintained (forced ventilation to each face)

Fig 15-14*c* shows the middle face ventilated by exhaust.

Another method of ventilating the dead ends of a haulage road, a stenton, and a manway is shown in Fig 15-15; the direction of air flow is indicated by arrows. The ducting is 500 mm in diameter; this method has been successfully used in many Donets Basin mines Its disadvantage is the series ventilation of coal faces, pack, and stenton, and chute.

A conical restricting fitting should be placed on the outlet from ventilation ducting to give the outflowing air a higher velocity; the air flow should be directed to the upper part of the face to blow down the gas or dust.

15-6. VENTILATION BY FANS

Fans for ventilating the faces of drivages and dead ends are generally fitted to the ducting described above, only occasionally to line brattices They are called *auxiliary fans*.

Auxiliary fans should be installed so that they do not suck in return air from the face.

Thus, for ventilating dead ends or faces which branch off from roads with a positive ventilating stream, the fan should be installed in the intake air at not less than 10 metres (with a high head, even further) from the point of contact with the dead end This rule is sometimes ignored in practice and the fan is installed either near the entrance to the dead end or at the point of contact with the intake airway. If the fan cannot be installed at the required point, it is necessary to prolong its ducting either on the suction side or on the forcing side, right up to the point where the air is sucked out

The fan can, in this case, be installed in the dead end itself

The intake to the fan should not exceed 70% of the air flow passing it in the intake airway, under the effect of the total mine head

As an exception, when cross-measure drifts and stone drifts are driven at the return airway horizon of a gassy mine, an auxiliary fan may be installed in the return airway provided that the methane content does not exceed 0 5%

The air at the face can be renewed in three ways: by forcing, exhaust, or a combination of both.

The advantage of forced ventilation (Fig. 15-16*a*) lies in the fact that the air passing out at a relatively high velocity from the end of the ducting continues its forward flow for some 10-15 m and, thus, sweeps out the gas mixture in the face more quickly than the

exhaust method, on the other hand, with forced ventilation, the shotfiring fumes, as pointed out, move along the roadway itself which naturally extends the re-entry period When the main road is not large enough for the fan to be located in it, the fan can be placed in the dead end provided that it has a sufficient length of ducting projecting into the main road; this method is used also with exhaust ventilation.

With exhaust ventilation (Fig. 15-16*b*) the ventilation ducting takes in only the gases in the immediate proximity of its end (1 to 1 5 m away from it towards the face) and the fumes thrown out

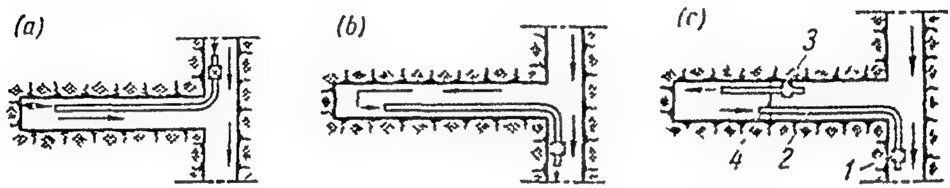

Fig 15-16 Methods of ventilating a dead end:
(*a*) forcing, (*b*) exhaust, (*c*) combined method

by shotfiring to a distance of about 25 to 30 m from the face; the gases which are nearer to the face inevitably stagnate; their removal and dilution takes place only by diffusion, that is, considerably more slowly than with forced ventilation; but because of the danger of damage to the ducting by pieces of rock thrown out during blasting it is inadvisable to bring it nearer to the face; it must be noted also that in a drivage through a very gassy seam, all the gas emitted from the walls passes into the face with the exhaust method of ventilation, finally, in this method the air is more strongly heated than with forced ventilation because throughout its journey into the face it is in contact with the walls of the roadway

The positive side of exhaust ventilation is that the roadway as a whole (except for 25 to 30 metres from the face) remains free of blasting fumes, and consequently there is no need to remove the men from the roadway throughout the period of ventilation, but the ventilation of the face itself takes longer.

With exhaust ventilation, the face, as pointed out above, is swept free of blasting fumes more slowly but the road as a whole is ventilated more quickly than with the forcing method A combination of the blower and exhaust systems brings in the advantages of both. The method includes basically an exhaust fan *1*; supplementing it in the face is a second (forcing or blower) fan (Fig 15-16*c*) The auxiliary fan *3* is set on a frame which moves forward with the face, and at the same time the ducting *2* is advanced For the forcing fan *3* it is convenient to use canvas ducting; *4* indicates a brattice

sheet or stopping, designed to prevent air flow from *4* to *3*; if this movement does occur, the output of fan *3* must be cut down. To eliminate this recirculation of the gases by fan *3* the output of fan *1* at the face should be 30% more than that of fan *3* without any sheeting or stopping, and 10% more if there is a sheeting or stopping.

The main fan *1* is installed near the mouth of the dead end and remains there.

If the main ventilation ducting is of canvas, the fan *1* must be placed at the face, it is sometimes possible to obtain satisfactory results through the stopping *4* merely with a length of blowing ducting without a fan, as near as possible to the face or using an injector, in an extreme case when there is exhaust ventilation ducting *2*, the face area can be quickly swept by a short blast of compressed air

Since in both types of combined ventilation, one (Fig 15-16*a*) or both fans (Fig. 15-16*c*) are not in intake air, these variants may not be used in gassy mines

Auxiliary fans produced in the Soviet Union use compressed air or electric power, and are radial-flow or axial-flow types

Compressed-air-driven fans are designed for work in metal mines or coal mines working steep seams, with compressed air supplies, they are also valuable for use in seams subject to sudden outbursts.

Compressed-air-driven fans of the axial-flow type are driven by a small-turbine mounted on the fan within the casing. They are described in Table 15-6

TABLE 15-6 **Technical Data of Compressed-Air-Driven Fans**

Description	Type of fan		
	ВП-3	ВП-4	ВП-5
Output, m^3/min	70-80	125-185	230-270
Pressure, mm water	50-40	150-80	90-60
Compressed air consumption, m^3/min	1 5	4 65	2-2 4
Compressed air pressure, atm	4-6	4	2-6
Weight, kg	19 5	65	45
Diameter of rotor, mm	300	418	500

Figure 15-17 shows guide vanes *1*; turbine rotor *2*; straightening vanes *3* at the outlet, turbine *4*; compressed air supply *5* to the turbine; outlet for the spent air *6*.

The characteristics of the ВП-3 and ВП-4 types are given in Fig 15-18 As Table 15-6 shows, the air flow from compressed-air fans is 15-100 times more than the consumption of compressed air,

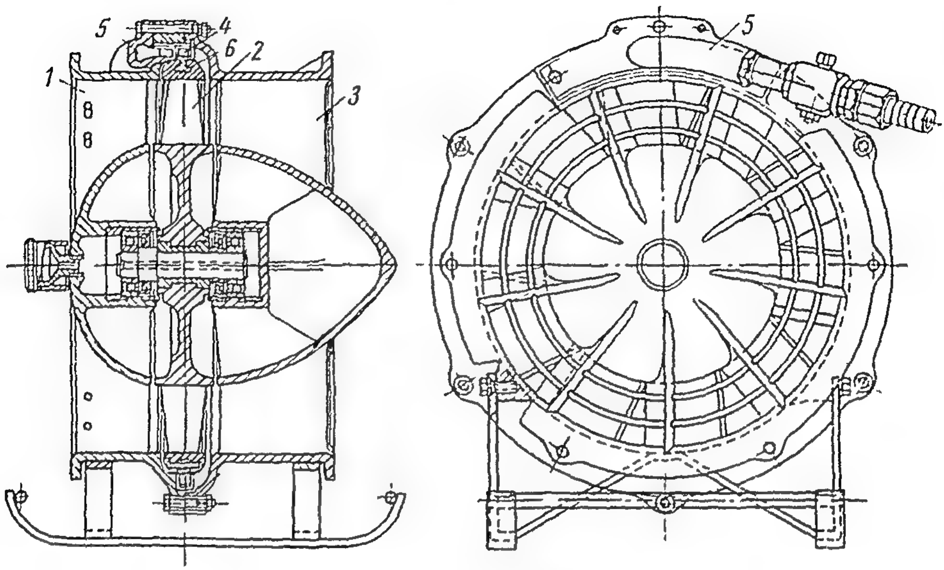

Fig 15-17 The ВП-4 fan

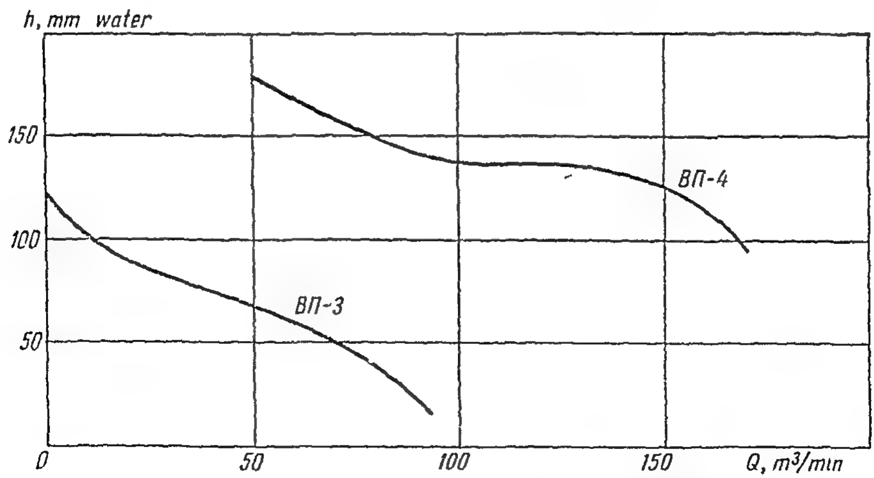

Fig 15-18 Characteristics of the ВП-3 and ВП-4 fans

at the same pressure It should be noted that at air pressures below those indicated in Table 15-6 the outputs and pressures from the fans diminish considerably

Of the two types of electrically driven fan, radial-flow and axial-flow, the first have been used underground but they are being replaced

by axial-flow fans, their main disadvantage is their large size and heavy weight. The usual type is the Sirocco, for example, the СТД-57 Nos. 4, 5 and 6 with outputs, under a maximum pressure of 240 mm, of 2.5, 3 3, and 4.5 m^3/sec; their efficiency is about 0 6; they weigh 110, 170, and 280 kg; the power consumption is 10.2, 15.2 and 17.4 kW.

Axial-flow fans are considerably more convenient (Fig. 15-19) and they are supplied together with the motor in the casing. They are compact and much lighter than radial-flow fans, which is extremely important underground. Figure 15-20 gives the characteristics of fans of the following types. ВМ-200, Prokhodka-400, Prokhodka-500, Prokhodka-600, ВМП-2-200, and ВМП-5-240 Some additional data on these fans are given in Table 15-7.

TABLE 15-7 **Data on Axial-Flow Fans**

Description	Prokhod-ka-400	Prokhod-ka-500	Prokhod-ka-600	ВМ-200	ВМ-300	ВМП-2-200	ВМП-5-240
Rotor diameter, mm	400	500	600	550	600	530	670
Maximum efficiency	0 65	0 65	0.65	0 4	0.5	0.6 approx	0 6 approx.
Motor power, kW	3.8	11	30	6 3	14	7 6	20.4
Weight, kg	140	265	470	200	315	190	247

Let us further mention the ВДМ-450 and the ВМ-600 with diameters of 450 and 600 mm, respectively, and with the following performance data: outputs 144-220 and 300-350 m^3/min at pressures of 250-130 and 200-160 mm; they weigh 205 and 258 kg, respectively Their maximum efficiency is 0 55 to 0 60.

Together with these comparatively small fans, more powerful fans have recently been used, particularly for shaft sinking: the ЦАГИ series B, with one or two stages, and a diameter of 1.2 m. the НГМ of 1.0 m diameter and the radial-flow fans Nos 9.5 and 12.5.

The performance data on the two-stage ЦАГИ fan at 1,500 rpm with $D = 1.2$ m are shown in Fig. 15-21 and those of the НГМ fan in Fig. 15-22.

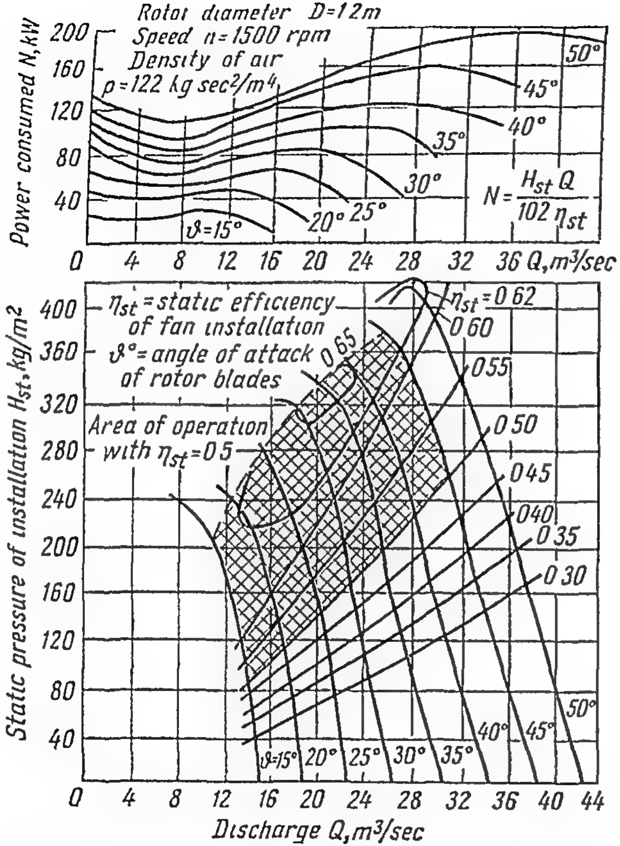

Fig 15-21 Characteristics of the ЦАГИ fan at 1,500 rpm

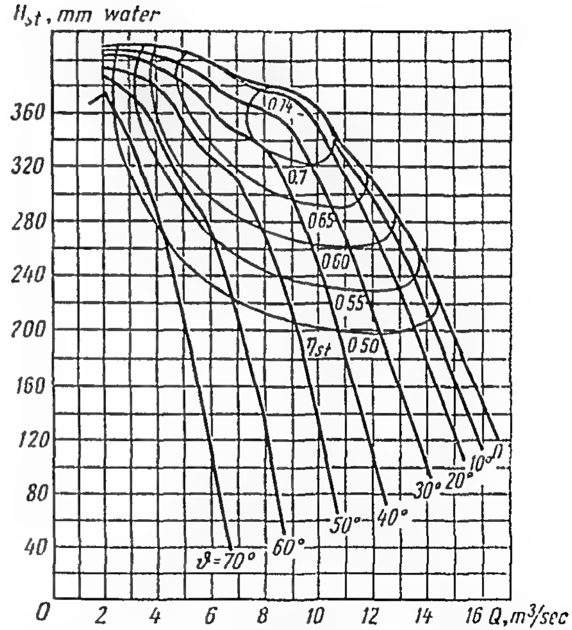

Fig 15-22 Characteristics of the ИГМ fan

The new radial-flow fans include the ВЦО-0.6/2970, the ВЦО-1/1485, and the ВЦО-1.2/1485 (the first figure indicates the diameter, and the second figure the fan speed); their outputs and heads are 300 m^3/min at h up to 600 mm; 700 m^3/min at h up to 420 mm; and 1,200 m^3/min at h up to 600 mm; their efficiency reaches 0 76 These fans are designed for ventilating deep shafts (1,000 m or more) and long drivages.

Let us also mention the so-called contra-rotating fans* first built in Britain; they are two-stage axial-flow fans with rotors turning in opposite directions resulting in a considerable reduction of the turbulence and swirl of the air flow downstream of the fan; their advantages include (a) a gently sloping characteristic; and (b) a high efficiency of 0 83, which changes little with change in flow rate.

For continuously monitoring the air supply to the face of the dead end and for switching off automatically the electrical machines in the face when the normal ventilation is interrupted, the PBB apparatus was designed by VUGI (the All-Union Coal Research Institute) The main element of this apparatus is a vane switch, the vane changing its position and breaking the contact of the switch when the air flow falls or ceases; after 2-3 min, a thermal time relay operates, breaking the contact in the control circuit of the magnetic starter, the power supply to the face equipment is then cut off. The vane switch is installed at the last section of the ventilation ducting

Instability. As is well known, axial-flow fans may stall in the transition of the operating conditions from the left downward branch of the characteristic at its hump and valley. Stalling is also possible in auxiliary axial-flow fans; after the ventilation ducting has reached a certain length, the operating conditions "break away" at the summit of the characteristic, falling to the valley, and the fan discharge drops The air leakages in this instance favour stability of operation.

15-7. CALCULATION OF THE AIR QUANTITY

The quantity of air flow needed for ventilating a dead end can be calculated on the basis of

(a) the amount of explosive used,

(b) the amount of dust formed,

(c) the gas quantity emitted

The method used for calculation is that which gives the largest quantity of air

---

* These contra-rotating fans have recently been built by Gipronikel (the State Design Institute for the Nickel Industry) in the form of the ВДМ-500-4 5 × × 2, when tested on the bench, this fan gave 2-2 5 m^3/sec at a head of 260 mm and an efficiency of 0 65

15-7 1 Calculation of the Air Requirement According to the Explosive Consumption

Depending on the method of ventilation, Q is determined from the following equations.

(a) *Forcing ventilation*

The equation of V N. Voronin

$$Q = \frac{7\ 8}{t} \sqrt[3]{AV^2}\ \text{m}^3/\text{min} \tag{15-4}$$

where t = time of ventilation, minutes
A = explosive consumption, kg
V = volume of the roadway to be ventilated, m^3.

Note In Equations (15-4) above and (15-17) (see below) implicitly under the root sign is the coefficient of aerodynamic resistance α, which is taken as equal to 0 00175, therefore air quantities calculated according to these equations in roadways with a coefficient α less than 0 00175 will be somewhat too high and in those in which α is more than 0 00175 will be somewhat too low

In ventilating long roadways, to reduce the re-entry period or the air quantity to the face, it is advisable to spray the face with water after shotfiring and to install water curtains across the return airway; in this case Equation (15-4) can be modified by a correction factor less than unity, for example 0 3 to 0 4

The equation of P I Mustel

$$Q = \frac{21\ 4}{t} \sqrt{AV}\ \text{m}^3/\text{min} \tag{15-5}$$

Note 1 In determining the volume V of the roadway equal to LS, in which S is its cross section in m^2 and L is its length, it must be remembered that in long roadways the concentration of harmful gases, as shown by experience, drops to the permitted level at a distance $L' = 450 \frac{A}{S}$ from the face (the formula of V N Voronin), this distance is about 300 to 500 m

Note 2 It is desirable for the distance l from the end of the ventilation ducting to the face not to exceed the zone of action of the free stream, and in particular (according to V N Voronin)

$$l \leqslant 0.5 \sqrt{S} \left(1 + \frac{1}{2a}\right)$$

where a is the flow structure factor of the free stream, equal to 0 06 for new ducting and 0 08 for old ducting, and S is the cross-sectional area of the roadway, m^2

Note 3 The air quantity should be checked for the minimum velocity which ensures effective ventilation, this velocity should not be less than 0 15 m/sec, but according to V N. Voronin

$$v_{min} \geqslant \frac{500}{d\sqrt{\alpha}}$$

where d is the average diameter of the props or a measure of the roughness in cm, and α is the coefficient of friction

(b) *Exhaust ventilation*

The formula of A.I. Ksenofontova

$$Q=\frac{6}{t}\sqrt{AS(75+A)}\ \text{m}^3/\text{min} \tag{15-6}$$

Note 1 Formula (15-6) is valid provided that the distance from the end of the ducting to the face does not exceed $l=3\sqrt{S}$; at larger distances the required air flow is considerably larger or, which amounts to the same thing, the ventilation time must be extended

Note 2 The average air velocity should not be less than 0.15 m/sec

(c) *Combined ventilation*

The formula of A. I. Ksenofontova for a forcing fan. If the distance l from the face to the stopping or to the exhaust ventilation ducting is less than 50 m

$$Q_f=\frac{15.6}{t}\sqrt{AV}\ \text{m}^3/\text{min} \tag{15-7}$$

where V is the volume of the roadway enclosed by the stopping, m³

If l is larger than 50 m, then

$$Q=\frac{112}{t}\sqrt{\frac{AV}{l}}\ \text{m}^3/\text{min} \tag{15-8}$$

When there is no stopping, the volume V can be taken as equal to that of the zone to which the fumes are thrown back, the quantity of air sucked into the exhaust ducting, as pointed out, should be 10 to 15% more than the output Q_f of the forcing fan if there is a stopping, and 25-30% more when there is no stopping. The output of the exhaust fan does not include the leakage into the ducting.

The fumes from shotfiring can be neutralized by a physico-chemical method. To speed the process of dilution of the fumes and consequently to reduce the time required for ventilation or to reduce the air required in the face, it has been suggested that the fumes be neutralized at the point of formation. Investigations undertaken in the ventilation laboratory of VNIIOMShS have shown that the following methods can render these fumes harmless.

(a) spraying a liquid chemical reagent with jets (0.5% solution of caustic soda), this neutralizes the nitrogen oxides;

(b) distribution of a powdered reagent in the form of slaked lime and Hopcalite (per 15 kg explosive 4 and 1.5 kg, respectively);

(c) passing air into the face with a fan through a filter containing a wad of Hopcalite and soda lime.

Industrial tests of these methods have confirmed that it is possible sometimes to halve the time needed for ventilation. They could be used for places difficult to ventilate, e.g. in long drivages with a considerable consumption of explosives.

15-7 2 Calculating the Air Flow on the Basis of Dust Formation

Part One of this book considered various methods of suppression of dusts formed underground mainly in drilling or blasting; in practice, however, it has been shown that some recommended methods of dust suppression do not reduce the dust to the required level; in addition, a positive air flow is needed at the point of dust formation. Only then is it possible to reduce the dust concentration to a permissible level The dust concentration of the air at the point of dust formation gradually falls because of·

(a) the removal of the dust suspended in the air by the ventilating current; the velocity of movement of dust particles is somewhat lower than the air velocity according to Professor N P. Neronov of the Leningrad Mining Institute;

(b) dilution by turbulent diffusion of the dust cloud by the intake air reaching the point of dust formation;

(c) the precipitation of the coarser dust particles helped by coagulation. A secondary process is the sticking of the dust particles onto the supports and the wall rocks; but it is possible that dust may be blown off the supports by the ventilating air.

We do not at present possess a generally accepted method of calculating the desirable quantity of air on the basis of dustiness Below are given some methods of calculating it employed in practice.

(a) *Calculation on the basis of the velocity of air flow.* According to V.V Nedin of the Krivoi Rog Geological Prospecting Institute, during exhaust ventilation and standard drilling using a single medium-size rock drill in hard quartzites and with the end of the ducting not more than 6 m from the face the air velocity should not be less than 0.6 m/sec; in similar conditions, but with forcing ventilation, the minimum velocity is 0.2 m/sec, with simultaneous wet drilling and loading into mine cars, the air velocity in a single-track road should be not less than 0.4 m/sec and in a double-track road rather less, but not below 0 2 m/sec

Similar values of air velocity have been obtained by the joint dust laboratory of the Mining Institute of the Kazakh Academy of Sciences and the Kazakh Mining Medical Institute, namely in horizontal drivages during rock filling, 0 2 m/sec, during drilling with two rock drills, 0 7 m/sec; ditto, with simultaneous rock filling, 0 9 m/sec; wet drilling with one rock drill, 0.4 m/sec; ditto, with simultaneous rock filling, 0 55 m/sec; at the breaker horizon, 0 8 m/sec, and in rise workings with air leg drills, 1.0 m/sec

(b) *Calculation based on the number of rock drills working simultaneously*. Usually, one rock drill is considered to require 1.0-1 2 m^3/sec of intake air.

(c) *Calculation based on the intensity of dust formation.* The amount of air is calculated from the formula

$$Q = \frac{A}{N_p - N_0} \qquad (15\text{-}9)$$

where A = index of intensity of dust formation (Tables 15-8 and 15-9)

N_p = permitted dustiness of the air in the return airway, assumed to be 400 particles per cm³

N_0 = dustiness of the intake air supplied to the working place.

TABLE 15-8 **Values of the Dust-Formation Index A in Various Drilling Conditions, Using Rock Drills HA-23 and HP-35**

Protodyakonov hardness	Ordinary wet drilling				Fully [illegible] drilling			
	HA-23		HP-35		HA-23		HP-35	
	A	A_{av}	A	A_{av}	A	A_{av}	A	A_{av}
Ore								
6-10	356-470	413	460-561	510	172-203	187	216-249	[illegible]
10-16	256-563	510	590-740	665	214-253	233	266-318	[illegible]
Ferrous rock,								
6-10	413-523	477	566-705	620	197-233	215	264-293	28[illegible]
10-16	632-790	710	880-1,120	1,000	315-368	342	423-498	46[illegible]

TABLE 15-9 **Values of the Dust-Formation Index A During the Work of Power Loaders and Scraper Loaders**

Type of operation and Protodyakonov hardness	Loading dry ore or rock		Loading wet ore or rock	
	A	A_{av}	A	A_{av}
Power loader ПМЛ-5, hardness				
6-10	471-573	525	172-204	183
10-16	440-545	492	147-172	159
Scraper loader, hardness				
6-10	328-378	353	188-226	207
10-16	269-332	300	140-163	151

The indexes of intensity of dust formation were obtained by V.V. Nedin from tests in the mines of Krivoi Rog; it can be assumed that with several rock drills working simultaneously, the index increases by the number of rock drills.

15-7 3 Calculation of the Air Quantity Required on the Basis of Methane Content

Observations on gas emission in development drivages have brought out the following conclusions:

(a) the gas emission takes place mainly from the fissured zone of the seam and in small volumes by seepage from depth in the coal seam;

(b) the drainage of the gas from the fissured zone is generally complete in 4-5 months;

(c) in long drivages. the gas emitted in the length driven in the previous month is equal to 50% of the total emission, that liberated in the preceding month is about 25%, that emitted in the month previously is 12% and so on, diminishing by a half with each preceding month;

(d) after the end of the drainage of methane, some quantity of gas still remains in the coal; it is, however, assumed that the coal in the fissured area is completely drained of gas, enabling us to ignore the gas coming out of the depth of the seam.

V L. Bozhko of MakNII recommends that the expected gas emission in development roads in gently sloping seams in the Donets Basin should be calculated in the following way

$$q_{gas} = \frac{kgA}{24 \times 60} \text{ m}^3/\text{min} \tag{15-10}$$

where k = coefficient which takes account of the excess gas emission of development roads, and is taken as 4

g = methane yield of the seam being worked, m^3/ton

A = 24-hour average output from the roadway, tons; it is calculated from the daily advance of the face with a width of 4 m.

Equation (15-10) is valid provided that the drivage is long enough, in particular that it has been driven for more than 5 months; if the time is shorter, for example 1, 2, 3 or 4 months, the gas emission must be determined from Equation (15-10) with the following correction factors: 0.5, 0 75, 0 87, and 0 93, respectively

In parallel roadways driven in gently sloping seams with coal pillars left between them, approximately 30 m wide, the total gas emission can be assumed equal to $2q_{gas}$.

In steep seams the total gas emission is equal to $1.5q_{gas}$.

In roadways driven singly, the quantity of methane emitted in 30 minutes after shotfiring can be high and will be approximately.

at driving speeds of 50 m/month	$q'_{gas} = 3q_{gas}$, m³/min
at driving speeds of 100 m/month	$q'_{gas} = 2q_{gas}$, m³/min
at driving speeds of 150 m/month	$q'_{gas} = 1.67q_{gas}$, m³/min

15-8. CALCULATION OF THE HEAD FOR LEAKING VENTILATION DUCTING

The head for leaking ventilating ducting can be calculated by various formulas:
from the formula for the arithmetically mean quantity of air

$$h = \left(\frac{Q_f + Q_0}{2}\right)^2 R \text{ mm water} \qquad (15\text{-}11)$$

from the formula proposed by V N. Voronin

$$h = Q_f \, Q_0 \cdot R \text{ mm water} \qquad (15\text{-}12)$$

from the formula

$$h = Q_f^2 \psi R \text{ mm water} \qquad (15\text{-}13)$$

for which the values of the coefficient ψ in relation to the value of the delivery factor $\eta = \frac{Q_0}{Q_f}$ are given by I A Shvyrkov (Table 15-10).

TABLE 15-10 Values of ψ

η	ψ	η	ψ	η	ψ
1.0	1.0	0.6	0.578	0.3	0.338
0.9	0.885	0.5	0.493	0.2	0.267
0.8	0.788	0.4	0.413	0.1	0.195
0.7	0.676				

From the equations proposed by V S. Vepiov of the Leningrad Mining Institute

(a) if $\eta > 0.35$, $h = Q_f^2 R (0.59\eta + 0.41)^2$ (15-14)

(b) if $\eta \leqslant 0.35$, $h = Q_f^2 R (0.72\eta + 0.125)$ (15-15)

The most accurate are Equations (15-13) and (15-14) which take account of the variability of the losses along the length of the ducting. Equations (15-11) and (15-12) are not recommended for values of η less than 0.5

The calculation of the head in leaky textile ducting was given in Section 15-5.

In the equations indicated, Q_f and Q_0 are the air quantities at the beginning of the ventilation ducting (at the fan) and at its end, and R is the resistance of tight ventilation ducting (without leaks).

15-9. VENTILATION OF LONG DRIVAGES

Long roadways (1 km or more) have to be ventilated in the driving of the long crosscuts or adits or haulage roads used in a retreating method of mining. Crosscuts are always driven singly and consequently they are ventilated only by a fan with ducting; shallow adits can be connected to the surface at appropriate intervals to shorten the dead end; haulage roads are ventilated by two methods (a) by driving parallel roadways, and (b) by ventilation ducting and a fan.

15-9 1 Ventilation of Paired Drivages

The ventilation is calculated in the following sequence

(a) the air flow required in the face is calculated, usually on the basis of the explosive consumption, occasionally on the basis of the airborne dust and with a check on the minimum velocity, which must be at least 0.15 m/sec;

(b) the air quantity needed at the beginning of the roadway, including all the losses through the stoppings in the stentons (Chapter 14);

(c) the necessary pressure is calculated;

(d) depending on the amount of this pressure and that which can be expected from the main mine fan, a decision is taken about whether to install an auxiliary fan (or fans) or to use the total mine head

15-9 2 Ventilation by a Fan with Ducting in a Single Drivage

There are three possible methods of choosing a fan for ventilating a long drivage:

(1) ventilation by one fan;

(2) several fans are installed together at the beginning of the drivage,

(3) several fans are distributed along the drivage.

In the first method the ratio Q_f/Q_0 is found from Equation (15-1), and then from Equations (15-13) and (15-14) the fan head is

calculated, from the values obtained for Q_f and h_f the fan is chosen. This method of ventilation is simple but may require the installation of a high-pressure fan of blower or aspirator type used in long drivages

In the second method, with several fans installed together, the fan is chosen as follows

(a) the value of the ratio $\eta = \frac{Q_0}{Q_f}$ is determined;

(b) the resistance of the ventilation ducting is determined with the air losses from it $R_{duc} = \psi R$ (the value of ψ is obtained from Table 15-10);

(c) the overall characteristic of the fans operating in series is drawn, i e. their ordinates are added; these characteristics are drawn for two, three or more fans;

(d) on the same diagram the characteristic of the ventilation ducting is drawn;

(e) the point of intersection of this characteristic with one of the overall fan characteristics is found at the discharge Q and this indicates the number of fans required

Ventilation by this method can be effective provided that

(a) the joints are made airtight in the most careful manner possible between the different fans as well as at the ports in the fan casings, if this is not done, the actual overall characteristic of the fans will differ greatly from the theoretical, and the eventual total head and output of the fans will be much less than that planned, as will be seen from the diagram obtained by V S Veprov in testing two and three fans connected in series (Fig 15-23)

The characteristics A, B, and C refer to the work, without losses, of one, two, and three series-connected Prokhodka-500 fans, the curves A', B', and C' refer to the same fans with air losses through leaks in the joints and casings,

(b) the fans are installed at some distance from each other, preferably not less than $10D$ in which D is the diameter of the ducting, or at the straightening guide vanes

Using this method of ventilation there is no need to install all the fans at once, at first one fan is installed, remaining single until the air flow delivered by it to the face becomes inadequate for ventilating within the required time; the length L of the ducting which corresponds to this condition is found in the following way from the known values of Q_0 and ψ and from the maximum value of the head which can be developed by the fan, we obtain from the equations

$$h = \frac{Q_f Q_0 L}{100} R_{100} \text{ or } h = \psi Q_0^2 \frac{L}{100} R_{100}$$

and the length L of the ventilation ducting

$$L = \frac{100h}{\eta Q_0^2 R_{100}} \text{ m} \tag{15-16}$$

where R_{100} is the resistance of a 100-metre length of airtight ducting in kilomurgs.

In the same way the point of installation of the second and subsequent fans can be determined. In each instance the graphical

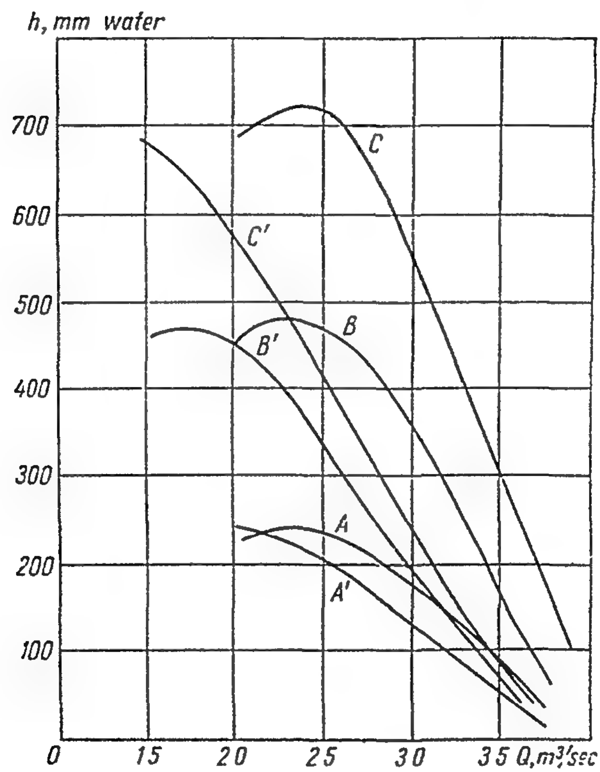

Fig 15-23 Characteristics of a single fan A and of two and three fans B and C in series without leakages The effect of leakages is shown by A', B' and C'

method must be used to check exactly what quantity of air will be delivered by the fan or fans.

If the fans are to be distributed along the drivage, their choice and location are arrived at in the following sequence:

(a) as for the first method, the value of the coefficient η is determined for the whole length of ducting; the air flow Q_f which must

be supplied to the beginning of the ducting is calculated, and then the value of the head which ensures the delivery to the face of the air flow Q_0; from these data a preliminary conclusion is made about what fans will ensure the necessary air delivery and how many fans should be installed (Items (c) and (e) of the second method above), further, the type and number of fans are more accurately determined, since with the distributed layout of fans the air losses will be smaller than with the first and second methods and consequently there will be both a lower output and a lower head from all the fans, this more accurate calculation is made as follows:

(b) by trial and error using in sequence different values of L, the length L_1 of the ventilation ducting is found at which the air flow Q_0 will be delivered to the end of the ducting at a given fan head h_1 and air intake Q_1;

(c) after obtaining the value of the flow from the first fan Q_1, by the same method of trial and error we find the location of the second fan and its output;

(d) the operation is repeated until the end of the ducting is reached

With this method of ventilation, the spacing between fans will be the less and their discharges will be the more, the closer the fan is to the start of the ducting, that is, to the intake air.

In a mine using this method of ventilation the following sequence is used one fan is installed, and it works alone until the ducting length reaches L_1; then the second fan is installed at the previously calculated distance L_2 from the first, and the two fans are used until the length of ducting calculating from the second fan again reaches L_1, after which the third fan is installed at the distance L_3 from the second fan and so on.

This method of calculating the ventilation of long drivages will be reliable provided that

(1) the air leakages at the fans and their joints to the ducting are negligible,

(2) the calculated coefficient of permeability k of the joints in the ducting is constant and not larger than the actual;

(3) the calculated resistance of the ducting is not greater than the actual and does not change with varying head.

Below is an example of the use of this method of calculating a long length of ducting

Example. The face of a road 1,500 m long requires 150 m³/min of air, choose the ducting and the fan

Solution 1 Ventilation with one fan We will assume that steel ducting 600 mm in diameter is used with the best connection possible in mine conditions and packing of the usual type, coefficient of permeability of the joints $k_1 = 1\ 11 \times 10^3$, which corresponds to 5% losses per 100 m, $l = 2$, $R_{100} = 3\ 0$

From Equation (15-1) we have

$$m=\frac{Q_f}{Q_0}=\left(\frac{kD}{3}\,\frac{L}{l}\sqrt{R}+1\right)^2=$$

$$=\left(\frac{1}{3}\times 1\ 11\times 10^{-3}\times 0\ 6\,\frac{1{,}500}{2}\sqrt{45}+1\right)^2=4\ 49$$

This value of m corresponds to a percentage of leakage $p = 100\ (1-\frac{1}{m}) =$ $= 100\ (1-\frac{1}{4\ 49}) = 77\ 8$ The fan output $Q_f = 2\ 5 \times 4\ 49 = 11\ 2$ m³/sec The fan head $h = Q_f Q_0 R = 2\ 5 \times 11\ 2 \times 45 = 1{,}260$ mm Using type M textile ducting we obtain from Table 15-4. $\frac{Q_f}{Q_0} = 2\ 085$ and $R = 28\ 8$, consequently $Q_f = 2\ 5 \times 2\ 085 = 5\ 21$ m³/sec, and $h = 5\ 21^2 \times 28\ 8 = 782$ mm

With steel ducting and high-quality jointing, $k = 0\ 034$

$$\frac{Q_f}{Q_0}=\left(\frac{0\ 034}{3}\times 10^{-3}\times 0\ 6\,\frac{1{,}500}{2}\sqrt{45}+1\right)^2=1\ 070$$

$$p=100\left(1-\frac{1}{1\ 07}\right)=7\%,\ Q_f=2\ 5\times 1.070=2.68\ \text{m}^3/\text{sec}$$

$$\text{and } h=2\ 5\times 2\ 68\times 45=302\ \text{mm}$$

Thus

(a) with steel ducting and the usual flanged connection

$$Q_f=11\ 2\ \text{m}^3/\text{sec}=672\ \text{m}^3/\text{min and } h=1{,}260\ \text{mm}$$

(b) with textile ducting

$$Q_f=5\ 21\ \text{m}^3/\text{sec}=313\ \text{m}^3/\text{min and } h=782\ \text{mm}$$

(c) with steel ducting and high-grade flanged connections

$$Q_f=2\ 68\ \text{m}^3/\text{sec}=161\ \text{m}^3/\text{min and } h=302\ \text{mm}$$

Using solution 1a, it is evidently impossible to use "Prokhodka" fans because none of them could give the necessary air flow (672 m³/min), this would only be obtainable with IIГM type fans, and to create the head (1,260 mm) it would be necessary to use four fans in series which evidently is not acceptable This shows that the use of steel ducting for driving roadways, using ordinary, although extremely carefully made joints, is not advisable

Using solution 1b for creating a head of 782 mm, two "Prokhodka-600" fans must be used

With solution 1c, i e using ducting with high-quality flanged connections, the conditions required ($Q_f=$ 161 m³/min and $h =$ 302 mm water) can be created by a single fan of the "Prokhodka-600" type or by two "Prokhodka-500" fans Since the head of 782 mm is too large for textile ducting 600 mm in diameter (see page 388) solution 1c is acceptable only

Solution 2 Ventilation using a distributed layout for the fans

It is possible to use textile ducting with two "Prokhodka-600" fans and the pressure from each fan will not exceed the maximum permissible value for this type of ducting (solution 2b)

In ventilating with steel ducting with improved flanged joints ($Q_f =$ $= 161$ m³/min and $h = 302$ mm) it is evidently possible to use two "Prokhodka-500" fans (solution 2c) One fan, however, will provide the necessary air

flow to the face until the ducting becomes 1,200 m long and the head reaches 236 mm, after which a second fan will have to be installed So as to reduce to some degree the leakages, this fan should be installed about 300 m from the first

After determining the fan locations, they should be slightly moved towards the intake, away from the face; this is done so as to avoid lengths of ducting being under pressure; this is always possible considering the inaccuracy of the values used in the calculations for ducting resistance and coefficients of permeability. In lengths of ducting under pressure (with the forcing method of ventilation), return air can be sucked back into the ducting, extending the time for ventilation and, in gassy mines, gassing up the face with all its consequences

15-10. VENTILATION OF SHAFTS DURING SINKING OR DEEPENING

Sinking shafts are ventilated by fans installed at the surface and connected to ducting lowered down the shaft One of two installations is possible for the ducting (1) suspension on ropes, and (2) permanent fixing on the shaft supports. The advantages of rope suspension are that the ducting can be raised before shotfiring instead of being taken to pieces, which is required by the second method; the ducting is more conveniently and rapidly erected on the surface as the shaft is deepened The second method has the advantages that there is no need to equip the shaft for hanging the ducting with hoists, ropes, pulleys, clamps, etc

In circular shafts with concrete lining, the first method is almost always used, i e suspension on ropes, in small shafts with timber lining, the preference is generally for the second method, i.e. fixing onto the supports

The diameter of the ducting is generally 400-600 mm, the length of one duct being 2-4 m, the joining of the suspended type of ducting is by flanged joints, and the permanently fixed ducting by couplings Suspended ducting is held between two suspension ropes (Fig 15-24) by laterally-restraining grips placed singly or in pairs on each ducting unit, not completely surrounding it, but sliding along the rope, and by supporting clamps solidly bolted to the rope and the ducting every 15-20 m or more; the clamps are fixed to the battens on the ducting (Fig 15-25) The vertical ducting is joined to the fan by a short curved length of tightly woven textile ducting. The hoist for raising the ducting has a capacity between 3 and 15 tons.

Rubberized canvas ducting replaces the last few lengths of steel ducting at the face, in this case instead of undoing or raising the whole length of ducting before shotfiring, it is enough to raise some 15-20 metres of canvas ducting

When the ducting is fixed to the shaft supports, each length carries a saddle of steel strip which is fixed by tie bolts or short chains

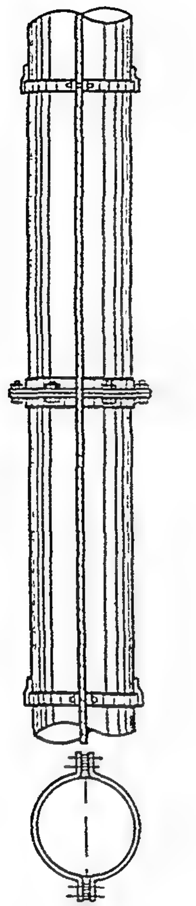

Fig 15-24 Suspension of ventilation ducting in shaft sinking

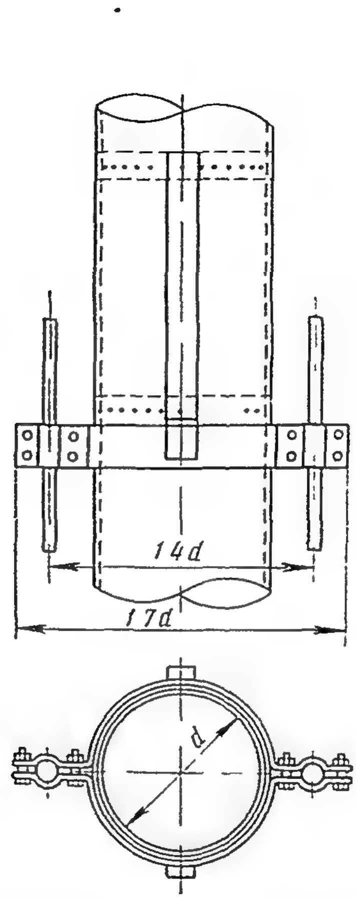

Fig. 15-25 Fixing of the laterally restraining grip to ventilation ducting

to anchors in the concrete (Fig. 15-26); in shafts with timber supports, the fixing is made without tie bolts (Fig. 15-27)

Apart from steel ducting, PVC-coated cotton ducting is also used, its weight is only one tenth that of steel

15-10.1 Methods of Ventilation

Of the methods of ventilation mentioned above, the *forcing method* is most frequently used Since the action of the free stream blowing out of the end of the ducting extends for not more than 20-25 m,

the distance between the ducting and the face should not be very much more than this, it is usually 30-40 m. Near the suspended stage in the shaft there are small zones of stagnant air which, however, does not greatly affect the ventilation

It is a disadvantage of this method that the shaft is fouled with fumes during the period of ventilation

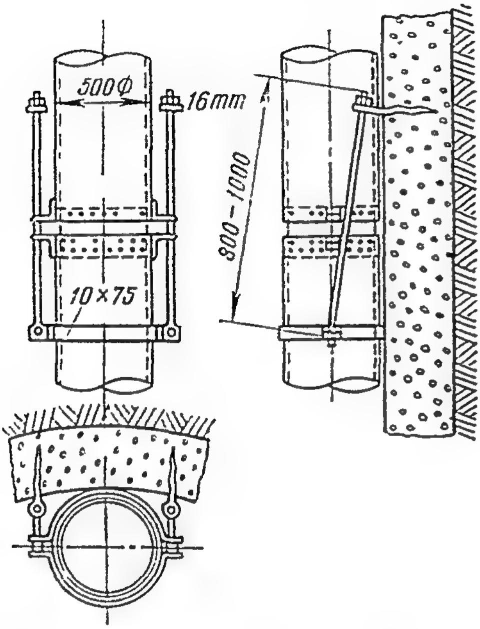

Fig 15-26. Permanent fixing of ducting to shaft walls

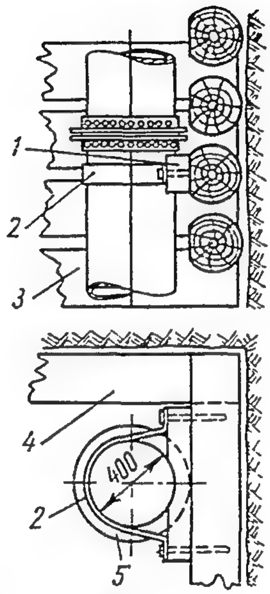

Fig 15-27. Fixing of ducting to a timbered shaft:

1—timber batten under ducting, 2—clamp made of strip steel, 3—ducting, 4—shaft timbering, 5—flange of the ducting

With the *exhaust method* of ventilation, as shown by the tests of N.P. Yakushin (VNIIOMShS), the air below the end of the ducting is under the action of the suction flow for a distance of only 1-1 5 m, below this the fumes from blasting are diluted with intake air but very slowly Another disadvantage of exhaust ventilation is that it is impossible to use textile ducting But its advantage is that the blasting fumes do not foul the shaft and are removed in the ducting

The efficiency of the *combined method of ventilating* depends on the ratio of the air flows Q_{exh}/Q_{for} The experiments of N P. Yakushin have shown that reliable ventilation takes place when this

ratio is above 1.2; with a smaller ratio there is a rising current of air in the shaft and the time required for ventilation is prolonged; because of the changing length of the ducting during sinking, the value of the ratio Q_{exh}/Q_{for} is continually changing, and a rising current may be formed, leading to fouling of the air in the shaft, which could poison the men working on the suspended stage. When the forcing fan has a constant length of ducting, and consequently the value of Q_{for} is constant, the ratio Q_{exh}/Q_{for} will depend only on the value of Q_{exh}. If by regulating the discharge of the exhaust fan it is possible to maintain constant the inequality $\frac{Q_{exh}}{Q_{for}} > 1.2$, then it would seem that there would be no reason against this method being used for rapid shaft sinking to reduce appreciably the time required for ventilation.

The forcing fan can be placed on the suspended stage (or under it) or at the surface. The first method is preferable because it does not require much forcing ducting in the shaft.

15-10.2 Calculation of the Air Quantity

The formulas for calculating the air quantity (Chapter 15) usually give high values, particularly for deep shafts. N. P. Yakushin modified the formula of V. N. Voronin which takes account of:

(a) the wetness of the shaft, as a result of which some of the fumes are dissolved in the water and do not have to be removed by ventilation;

(b) the fact that with forcing ventilation the air losses help to dilute the fumes by the upward movement of the gas mixture in the shaft.

This formula (only for forcing ventilation) takes the form

$$Q_f = 0.13 \frac{S}{t} \sqrt[3]{\frac{k_w A L^2}{S\, m^2}} \text{ m}^3/\text{sec} \qquad (15\text{-}17)$$

TABLE 15-11 **Values of the Wetness Factor for Shafts, k_w**

Description of shaft	Wetness factor, k_w
Dry (up to 1 m³/hour of water) and wet for not more than 200 m in depth	0.8
Wet (up to 6 m³/hour) and deeper than 200 m	0.6
Wet (from 6 to 15 m³/hour), condition like rain	0.3
Wet (over 15 m³/hour), torrential downpour	0.15

where $m = Q_f/Q_0$ = coefficient of air loss, determined in Sec. 15-5
k_w = factor indicating the degree of wetness of the shaft; the value of this factor can be obtained from Table 15-11.

See also note on page 403.

15-10 3 Ventilation in the Sinking of Two Shafts Simultaneously

When a pair of shafts are sunk, they are ventilated in the usual way to a certain depth by surface fans; they are then joined by a crosscut, and the ventilation layout can be changed to the method

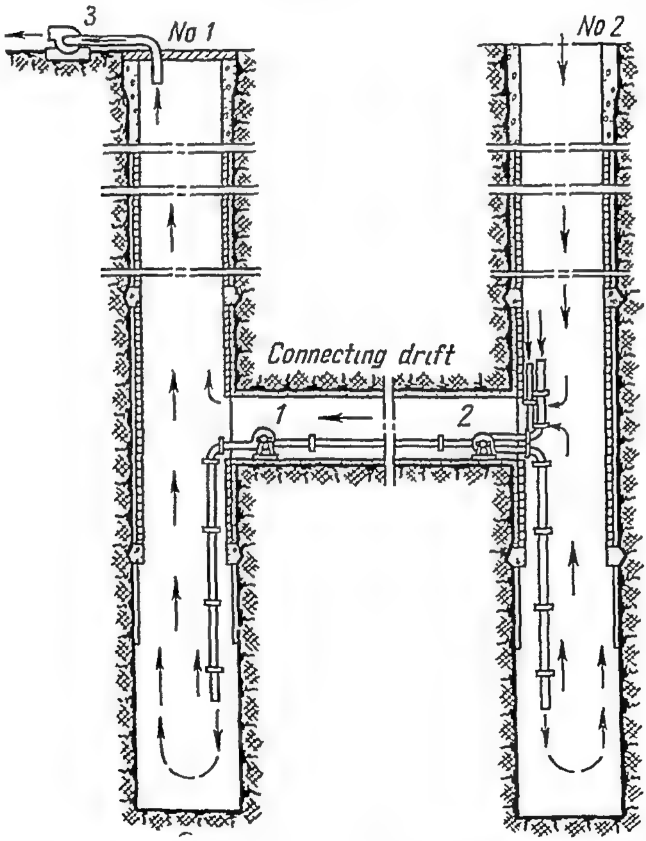

Fig 15-28. Ventilation of two shafts being sunk simultaneously

indicated in Fig. 15-28; two fans *1* and *2* independently ventilate the two faces being sunk (it is also possible to install a single powerful fan); to improve the ventilation of the left-hand shaft, the fan *3* installed at its surface can then be reversed.

15-11. USE OF COMPRESSED AIR FOR VENTILATION

In mines which use compressed air, the latter can be used for ventilation in: (a) injectors, and (b) fans.

Mere release of compressed air is not advisable because it is inefficient and expensive because of the high cost of compressed air as compared with electricity.

In addition, as pointed out above in Part One, the use of piston compressors may result in the release of poisonous gases containing carbon monoxide into the face, which can poison the men breathing it; compressed-air-driven auxiliary fans and injectors are less dangerous because the compressed air is diluted by not less than 10 times

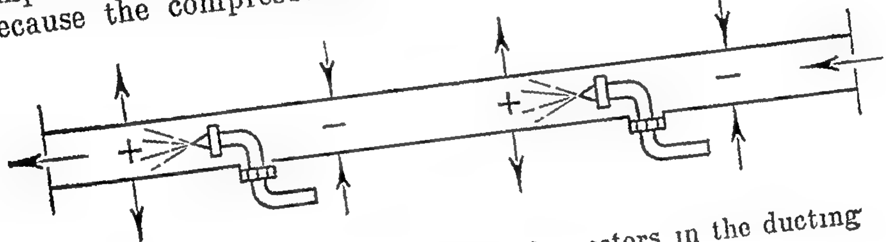

Fig 15-29 Diagram showing layout of injectors in the ducting

its volume of intake air. It is worth mentioning that in the mines of Krivoi Rog a remote method of air analysis has been used in the driving of raises.

An *injector* is a ventilation device using the power from a compressed air jet (Fig. 15-29); in front of the injector the air is forced forward by the velocity head of the air from the jet; behind the jet is an area of low pressure, into which the air is sucked from outside.

The simplest injector is connected to a compressed-air pipeline through a T-piece by a small pipe with a nozzle at the end; the nozzle is screwed into the pipe.

An improved injector is shown in Fig. 15-30; it is installed inside a Venturi tube (Fig 15-31). The dimensions of the nozzle and the Venturi are standardized The nozzle *1* (Fig. 15-30) is made from circular brass bar, 35 mm in diameter, drilled to a double cone shape, slightly widening at the outlet, and drilled to 27 mm at the opposite end where the nozzle has an external thread (for a length of 30 mm) on which a coupling *2* with a cross *3* is screwed, which both connects the nozzle to the compressed-air pipe and fixes it inside the ducting.

Many ways of calculating injectors exist, but they are all complicated and their use for the design of auxiliary ventilation can hardly be recommended. One of these calculation methods is described below in brief form.

Let the air flow required from the injector be 0.4 m³/sec in a duct 300 mm in diameter ($S = 0\ 07\ m^2$) for a distance of 50 m; the pressure p of the compressed air is 5 atm gauge. The frictional resistance

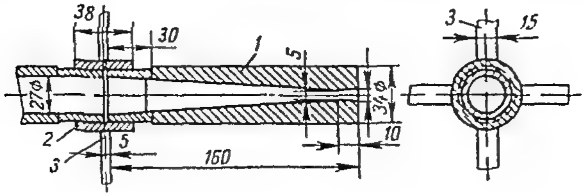

Fig 15-30. Injector

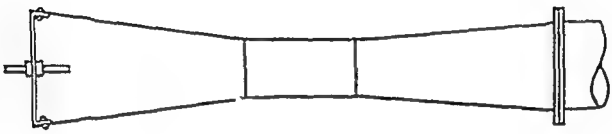

Fig 15-31 Venturi tube, suitable for the installation of an injector

of the duct (see Table 15-12) is equal to 60 kilomurgs; the air velocity at the outlet $v = 0.4/0.07 = 5.7$ m/sec; the pressure loss from friction $h = RQ^2 = 60 \times 0.4^2 = 9.6$ mm; the velocity pressure at the outlet $h = \frac{v^2}{2g}\gamma = \frac{5\ 7^2}{19\ 62} \times 1.2 = 2$ mm; the *total head* is $9\ 6 + 2 = 11.6$ mm The necessary force in this jet is obtained from the equation: $F = hS = 11.6 \times 0.07 = 0.81$ kg.

TABLE 15-12 **Data on Compressed-air Jets**

Air pressure, kg/cm²	Compressed-air consumption per mm², litre/sec	Force of jet per mm², grams
2	0 57	29
3	0 76	42
4	0 95	55
5	1 14	68
6	1 33	81
7	1 52	94
8	1 71	107

From Table 15-12 we find that at a pressure of 5 kg/cm² the force of the jet per mm² of the cross section of the nozzle is 68 grams,

and consequently the nozzle cross section required is 810/68 = = 12 mm², and d for the nozzle is 3.9 mm. The compressed-air consumption will be $Q = qs = 1.14 \times 12 = 13.7$ litres per sec = $= \frac{13.7 \times 3{,}600}{1{,}000} = 49.2$ m³/hour. The results of the calculation agree well with tests on injectors.

In long ducting several injectors should be placed in series.

The Safety Regulations do not indicate at what distance the injector intake must be placed from the return airway; remembering the small suction which is created by the injector, we could assume this distance to be 4-5 metres, instead of the 10 metres required for fans.

The joints between the sections of ducting should be made tight, and the compressed-air pressure should be not less than 4 atmospheres.

15-12. VENTILATION THROUGH DRILLED HOLES

Drivages can often be effectively ventilated through boreholes, for example:

(1) in shallow deposits in the topmost horizon, or in the first development of a horizontal deposit, the roadways as they advance can be connected by boreholes to the surface; to increase the air flow along the boreholes, small fans are installed on them;

(2) when a haulage road and return airway are driven simultaneously, to ease the ventilation, the roads are joined by boreholes, to which ducting with a fan is connected;

(3) when roadways are driven together, a large-diameter borehole is driven between them, instead of a stenton.

Borehole ventilation is calculated in the following way:

(1) the friction factor (according to A. Dzasokhov) can be taken for unlined boreholes to be 0.00042, and for cased holes to be 0.00029; it must be remembered that a cased hole is narrower than an uncased hole;

(2) when the resistance of the borehole is more than 50 kilomurgs, the air flow which can be given by the fan installed at the borehole is calculated from the equation

$$Q = A \frac{u}{L} \sqrt{L} \text{ m}^3/\text{min} \tag{15-18}$$

where $A = 7.1$ for cased boreholes and 5.8 for uncased boreholes
u = rim speed of the fan rotor, m/sec
L = length of the borehole, metres

The fans can be installed either at the foot or the top of the borehole; in the former case, to reduce the loss of velocity pressure at

the outlet, the borehole should be fitted with a small evase or diffuser on the surface

To increase the air flow two boreholes can be driven close together and they can be joined to the same fan.

15-13. SPECIAL FEATURES IN VENTILATING DEAD ENDS IN OVERHEATED DISTRICTS OF COPPER PYRITE MINES

Because of the oxidation of copper pyrite ore, the air temperature in the development drivages of copper pyrite mines is often higher than permitted by the regulations Investigations of the ventilation of these workings have shown that to reduce the air temperature in dead ends, special measures are needed (see p. 519).

CHAPTER 16

VENTILATION STRUCTURES

16-1. GENERAL

The suitability of the ventilation structures used in mines has a considerable effect on the overall head of the mine as well as on the quantity of air reaching the faces Poor fan drifts, air crossings, sharp unsmoothed bends, etc , often absorb some 30 millimetres of water gauge. Large quantities of air are short-circuited to the return airways or to the atmosphere through faulty fan drifts, or buildings over the upcast shaft, air locks, doors, and by-passes to inclines.

Ventilation structures should have both minimum aerodynamic resistance and minimum leakage. However, because of the specific working conditions—the presence of rock pressure, the temporary character of most structures, and restriction of space—underground ventilation is often provided without satisfying these requirements

Ventilation equipment and structures can be divided by function into two groups as follows.

(1) structures for passing air (fan drifts) and for changing the direction of air such as air crossings and ventilation ducts;

(2) structures for shutting off air — stoppings, doors, trapdoors, valves, by-passes at the foot of inclines, and the buildings over upcast shafts.

Structures of the first group should be streamlined so as to achieve the minimum resistance of the air through them. Structures of the second group require maximum airtightness.

16-2. DESCRIPTION OF VENTILATING STRUCTURES

1. **Fan Drifts.** Fan drifts take all the air coming out of the mine or going into it and therefore the two requirements mentioned above—airtightness and minimum aerodynamic resistance—should be satisfied to the maximum. Air leakages in inferior fan drifts sometimes reach 10-15% of the fan throughput; in other mines the fan drift has such a high resistance that it may exceed that of the fan, considerably reducing the fan output and resulting in a useless waste of power.

The fan drift should satisfy the following requirements:

1) The cross section should be adequate to keep the air velocity in it down to 10 m/sec maximum (according to the rules for mine operation, up to 15 m/sec), it should be remembered that the cost of building a wide enough fan drift will always be repaid by the saving on the power cost, the number of bends should be as low as possible

2) The fan drift should be as short as possible; the fan should be as near as the surface planning allows, to the collar of the upcast shaft.

3) The junction of the fan drift with the shaft should be in one of the following forms (a) an incline; (b) a smooth bend; (c) a bend with the inside of the corner chamfered.

A sudden restriction to the flow together with a 90° bend is not allowed If the ventilation shaft is not used for hoisting, guide vanes should be installed at the bend (Chapter 8) and the dead end above the drift should be closed.

4) The drift should be as smooth as possible (for example, plastered) and should not be encumbered; if bends are necessary, they must be smooth or provided with guide vanes.

5) The centre line of the fan drift should be in the same plane as that of a single-inlet radial-flow fan and perpendicular to the centre line of a double-inlet fan To avoid lack of symmetry in the air flow onto the fan blades, the drift should have no bend in front of the fan

6) The bend from the drift to the inlet of a radial-flow fan should have a rounded internal edge

7) If the drift is above ground, it should be heated to avoid freezing up

8) The drift should have a manhole with two strong trapdoors so that it can be inspected, and cleaned, and so that the air flow can be measured

9) When the drift receives much water from the air flow, and the water flows back into the shaft (sometimes considerably increasing its resistance), the drift should be provided with a cross gutter to catch the water; the water can be removed by a small pump.

Fan drifts have been known to fill with ice in winter (with a large short circuit across the shaft collar), or to be blocked with coal dust (in dusty mines the dust is dropped in the fan drift with the condensing water)

Observations in many mines in various areas of the USSR have shown that the pressure loss in fan drifts results from inferior construction or from restrictions due to ice or dirt; it may reach several tens of millimetres and has individually exceeded 100 mm.

For the resistance of fan drifts see Appendix 4. Tentatively, the following values of resistance can be assumed: a good drift of large cross section, $R = 0.52$ to 2.0 murgs, a good drift of average cross section, $R = 5$ to 10 murgs; the pressure loss in large- and average-sized drifts should not exceed 10-20 mm.

The intakes of forcing fans should be built, if possible, without bends or with guide vanes at the bends; the intake in the form of a short horizontal duct changes over into a vertical well at the top of which is built a cabin with louvres for keeping out the snow, this has a fairly large resistance to the air flow; according to one of the authors' measurements at a metal mine at Krivoi Rog, the pressure loss at this intake was about 24% of the total mine head.

Figure 16-1 shows the fan drift of an axial-flow fan with a lengthened diffuser or evase. To reduce the considerable noise from these fans, which interferes with people working on the surface and disturbs the people living nearby, it has been proposed·

(a) to build at the outlet from the fan drift a silencer in the form of walls made of hollow, sound-absorbing slabs filled with wadding or cement-slag blocks; these silencers increase the size of the outlet and somewhat increase its resistance;

(b) to build a sound reducing evase and fan casing, as shown in Fig. 16-1, in the form of brick walls, filling the space between them and the metal casing of the fan with sand, above, instead of sand, a layer of felt and sawdust is placed.

2. The Structure Closing the Collar of the Shaft. The collar of the shaft at which the fan is installed, whether exhaust or forcing, should be covered to avoid short circuits between the fan and the atmosphere.

The following methods of making the shaft airtight are used

(1) in shafts and small pits not used for hoisting—airtight non-openable floors or stoppings covering the whole cross-section of the shaft;

(2) in shafts and inclines used for hoisting men and minerals—air locks, special gates, or an airtight shaft building.

Non-opening floors over small pits are built in the form of a deck consisting of two layers of squared timber, or thick boards, tongued and grooved, and with a layer of tarred paper, or sheet steel between them. Stronger decks are built of steel joists and rails; sometimes the collar is covered with a brick vault. These non-opening decks are covered with a layer of clay about 30 cm thick. It is advisable to leave an opening in the deck which can be tightly closed by two trapdoors to enable men to pass in or out.

At gassy mines, to reduce the pressure on the fan in case of an explosion of gas or dust underground a vent should be built into the fan drift wall, not nearer than 7.5 m from the shaft. The vent

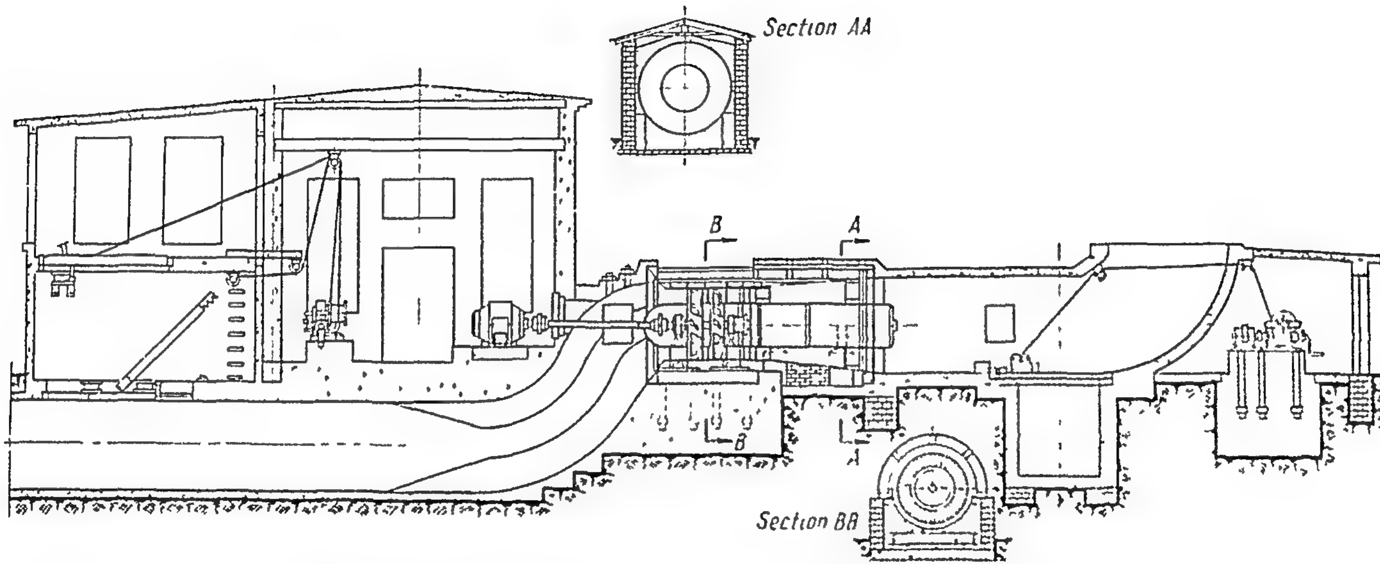

Fig 16-1 Layout of fan installation with a long diffuser

has only a slight resistance to the pressure and opens to the atmosphere when the pressure in the fan drift increases; this vent should be located not nearer than 7.5 m and not further than 30 m from the fan; the vent should be between the fan and the possible explosion With careful supervision, the quantity of air leaking through non-openable decks can be reduced to 1-2%; in fact it is often 5% or more.

3. **Air Locks.** Air locks usually have two doors fairly close to each other; their function is to allow men and mineral to pass through without interrupting the ventilation; when one door is open, the other must be closed. Air locks are built in the buildings

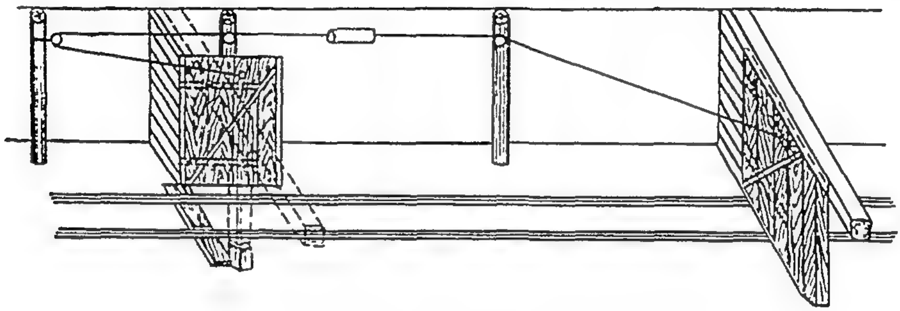

Fig. 16-2. Interlocking of air doors

over ventilation shafts and at the collars of ventilation drifts; in addition, they are necessarily built underground at places where there is much traffic, when frequent opening of a door can disturb the ventilation; the space between two doors must be equal at least to the length of a train of mine cars.

Methods of making doors in air locks tight are described above (Chapter 14). Since the quantity of air leaking through the doors is inversely proportional to the square root of their resistance, the second door reduces the leakages by a factor of about 1.4 and the second and third together reduce it by a factor of 1 7. To reduce the resistance of the bend from an incline into the fan drift, the inner door of the air lock should be located as near as possible to the fan drift so as to eliminate the re-entrant angle at the dead end

The air locks in the buildings over the ventilation shafts, used for hoisting minerals, are sometimes mechanically operated (by compressed air or more often by electricity) and with interlocks to prevent the two doors opening simultaneously (Fig. 16-2). The doors are usually of the lifting type Figure 16-3 shows a compressed-air-operated door with a counterweight opposing the compressed-air cylinder. An air lock with interlocks is shown in Fig. 16-4; the doors are connected to ropes which are wound onto the opposing

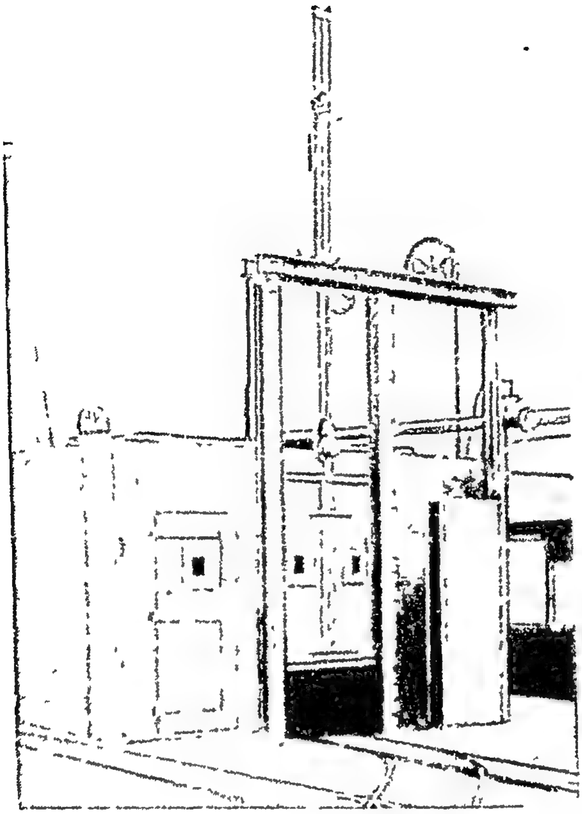

Fig 16-3 Equipment for closing air locks

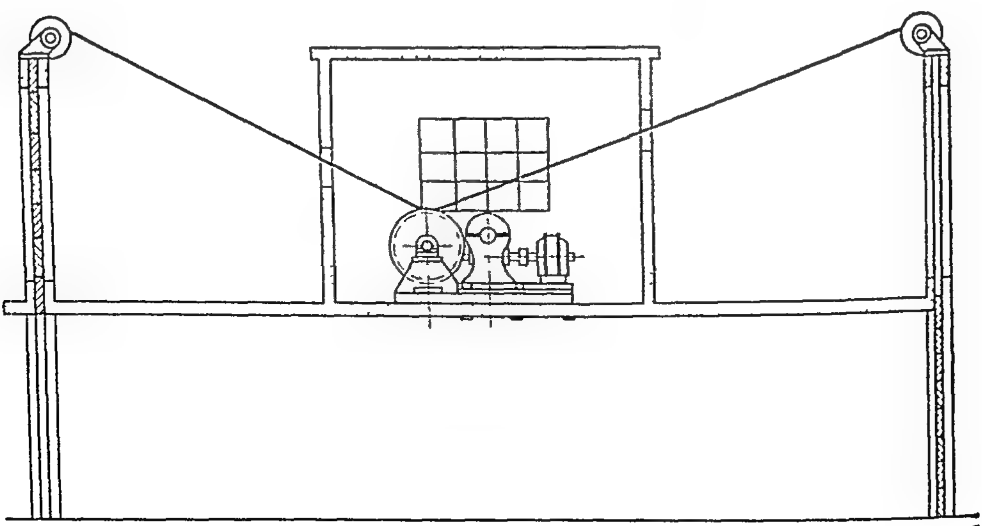

Fig. 16-4. Interlock for doors in an air lock

sides of two sheaves turning freely on the same shaft The sheaves have a small projection on one side. When the shaft rotates, a pin on a coupling keyed to the shaft carrying the sheaves comes into contact with the projection on the sheave and turns it, one rope then winds onto the sheave and the other unwinds, one of the doors being raised, while the other is lowered. The couplings are so designed that one of the doors is always down. The air lock is served by one man.

Automatic air locks are built so that a moving mine car turns a lever with its axle, starting an electric motor, which raises one door and lowers the other, allowing the required number of mine cars to pass through.

To allow only men to pass, rotating air locks are sometimes used (Fig. 16-5).

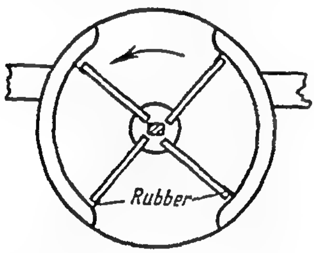

Fig 16-5. Swivelling air lock

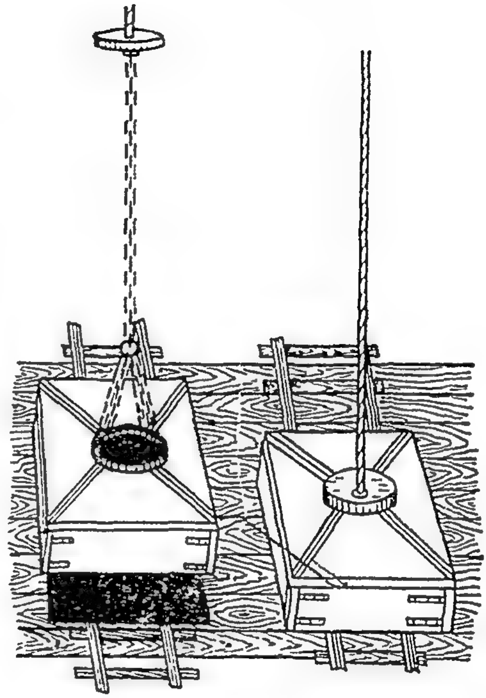

Fig. 16-6. Lifting doors over a shaft

When the mine has a large pressure, the doors of air locks are forced so hard against the frame that it is difficult to open them. To make opening easier, holes are cut in the doors and closed by small shutters; when the shutter is opened, the pressure is equalized on the two sides of the door. The shutters can be liftable and are opened by handles on each side of the door

4. **Shaft Doors.** When the building over the ventilation shaft is not airtight, the shaft is equipped with special doors or shutters to form a simple air lock enabling men and minerals to be hoisted without disturbing the ventilation, i. e without short circuits between the atmosphere and the shaft They consist of lids which cover the opening for the cage at the banking level of the shaft. The shaft should be airtight from the surface at the banking level.

The construction of these doors is shown in Fig. 16-6. They are made of steel (thickness averaging 4-6 mm) or timber strengthened with steel angles or flats; timber doors distort less than steel ones and can be more easily fitted to the shape of the banking floor but they are heavier The door, as the drawing shows, is opened by the chains on the cage being hoisted, while to ease the lifting of the door, it is provided with a small lid which is opened before the door to equalize the pressures on each side of it.

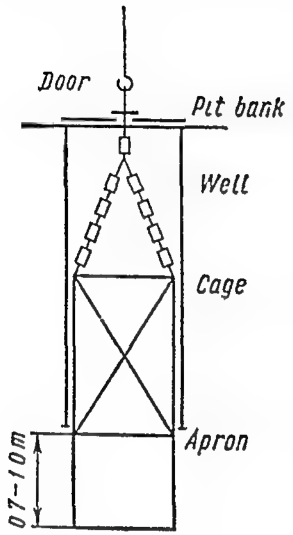

Fig 16-7 Diagram of well built for a cage at banking level

To reduce the short-circuiting of air when the door is lifted at each hoist of the cage, a small well is built (Fig. 16-7) for each cage in which it passes like a piston, with minimum gaps, it is long enough for the bottom of the cage to just pass into the well during hoisting of the door. The bottom of the cage is also provided with an apron, 0.5-1 m long; when there is a small overwind, the apron prevents the air passing into the shaft.

The shutters (or doors) used with rail guides are slotted, or the guides are cut, and on the roof of the cage on each side is a small frame with guide shoes to transfer the cage across the gap in the guides.

Disadvantages of these doors are that they put extra tension on the hoisting ropes, the air short-circuited may reach 20% and averages 12% (with careful supervision this can be reduced to 7-8% or less), and the door is destroyed when the cage is overwound

Their advantages include low cost, simplicity of supervision, and ease of repair

5. Airtight Shaft Buildings. Instead of the doors or shutters, a ventilation shaft can be closed by an airtight shaft building with air locks for men and minerals to pass through

Sometimes shutters are provided in addition to the airtight building

The part of the building under pressure can be the whole building if it is not large (Fig 16-8) or only one floor or part of a floor, when only the part of the building next to the shaft is under pressure (Fig 16-9), the doors are usually of the lifting type (Fig. 16-3). Sometimes the shaft is made airtight at the pit bottom, and lifting doors are not used If hoisting is by skip, the ventilation shaft is made airtight by means of·

(a) a layer of coal, the depth of which depends on the coal size and on the pressure drop of the mine (usually 2-4 m),

(b) an air lock with steel gates.

Usually method (a) is used for making airtight at the surface and method (b) at the pit bottom. Special rotating tipplers are also used which are placed in an airtight casing.

The unloading devices for skips are made airtight usually with rubber washers; but these washers wear very quickly and it has

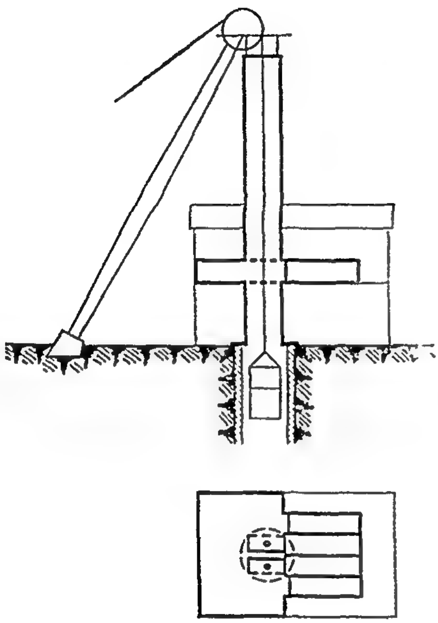

Fig 16-8 A building above a mine shaft, partly or entirely airtight

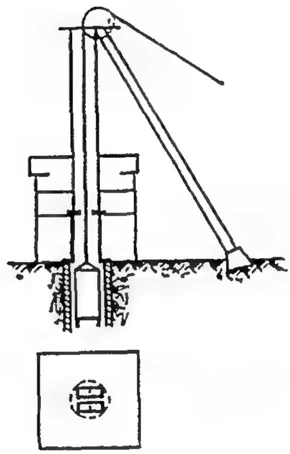

Fig 16-9 A building over a mine shaft in which only the part directly above the shaft is made airtight

been proposed to replace them with labyrinth packings which do not wear; after rubber washers had been replaced by labyrinth packing, the air leakages were reduced to below one fifth of their former level.

6. **Structures for Changing the Direction of Air Flow.** The experience in preventing underground fires and explosions has shown that mine rescue work can often require a change in the direction of the underground air flow, in fact its reversal. When the air flow is reversed, an exhaust fan becomes a forcing fan, and vice versa

The flow is reversed usually by a by-pass drift and doors (Fig. 16-10*a*. *b*, *c*); when the flow is reversed, the door *1* is lifted

Doors are the most important mine ventilation structure, fulfilling the same function as stoppings; if doors are leaky, particularly with a high pressure difference across the stopping, large quantities of air can be lost Air doors between shafts have been known to lose 20-30% of the mine air flow by short-circuiting.

A door left open can completely interrupt the ventilation of a whole district of a mine, and a door left closed when it should be open can also deprive a face of air A door located in the wrong position, for example, in a seam subject to spontaneous combustion, can cause a fire, because of the pressure difference in the seam, created by the door, the air will pass through cracks in the seam and cause self-heating in the coal

Doors are made of steel or timber strengthened with steel Steel doors are built in the drift connecting the donwncast to the upcast shaft, and in those places where a short circuit (caused by an open door) can cut off the ventilation of a section of the mine, which forms a separate take; the thickness of such doors should be not less than 40 mm if they are made of timber, and 3 mm if steel is used.

The construction of timber doors is shown in Figs 16-11 to 16-14, Fig 16-11 shows a door strengthened by strip steel, Fig 16-12 shows a two-leaf door; Fig. 16-13 shows a swinging door, Fig 16-14 shows a door with a regulator.

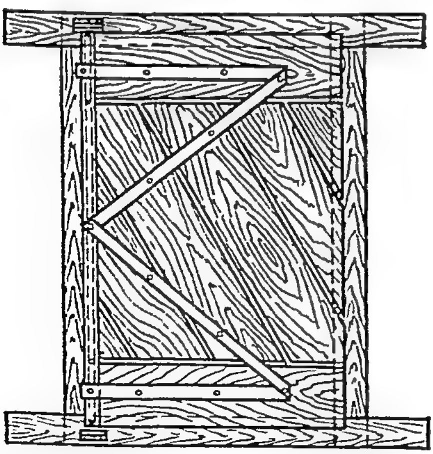

Fig 16-11 Timber air door, stiffened by steel strip

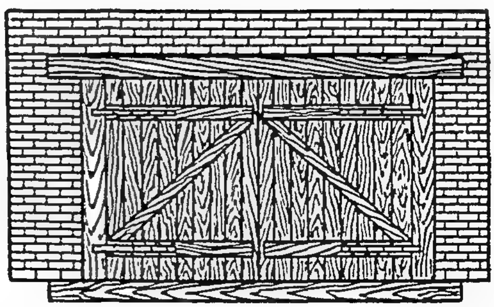

Fig. 16-12. Two-leaf air door

Timber doors are built either from one or two layers of boards nailed to two cross ledgers, or the ends of the boards are housed into a mortise cut in the top and bottom ledgers, or finally they may be framed all round, the boards are nailed together, tongued and grooved or simply carefully jointed, so that

doors shall not sag, they are strengthened with one or two braces housed into the boards Doors nailed together from two layers of boards have the layers at right angles to each other or at 45°, greater airtightness is achieved by putting in a sheet of tarred paper or brattice cloth between the layers; also a piece of

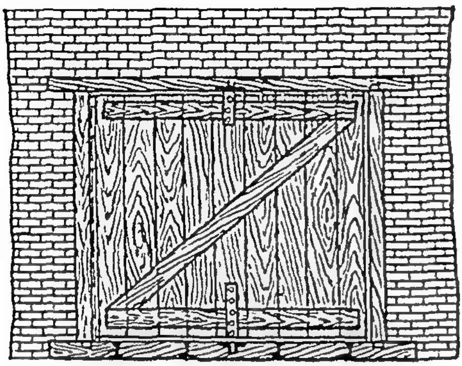

Fig 16-13 Centrally pivoted air door

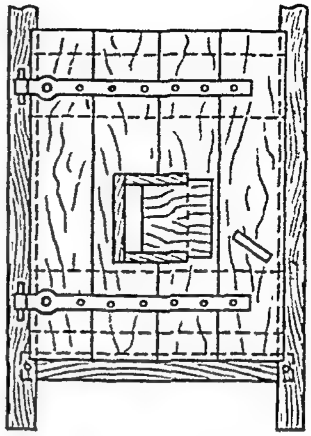

Fig 16-14 Air door with regulator (scale door)

brattice cloth is nailed to the bottom of the door on the intake side, in addition, strips of felt are sometimes nailed round the door to fit it better to the door frame.

As Fig 16-11 shows, mine doors are usually hung in such a way that they lie on the door frame, being forced on to it by the air pressure, if a rebate of the usual type were to be cut in the door so that the door passed into the frame, under heavy rock pressure the door would be wedged in the frame and could not be moved out of it

In mine workings with steady rock pressure, ordinary doors are sometimes built, hanging in a complete door frame with a threshold These doors, if properly fitted and hung, allow little or no air to pass

Doors are supplied with handles and latches which open from both sides If the pressure difference between the two sides of the door is considerable (it should be remembered that 1 millimetre of depression corresponds to one kg/m^2), the door is difficult and sometimes impossible to open by one man. When this is so, the door is opened more easily by a special bent lever (Fig 16-15*a*) The lever with its arms *1* and *2* hinges on the pivot *3*, to which is fixed a steel strip *4* with a roller, when the lever turns, by lowering the arm *1* or lifting the arm *2*, the roller presses on the door *5* and opens it. A simpler lever is shown in Fig 16-15*b*, when the lever is pulled

its short end bears on the frame and the door is slightly opened, equalizing the pressure. Small holes are also sometimes cut in doors and provided with a shutter, particularly in doors with a large pressure difference These are generally provided at an air lock The shutter opening should be large enough to allow a man to crawl through a door in case the second door has been accidentally left open; otherwise, even if the shutter is opened, it will often be impossible to open the door.

So as to avoid doors being accidentally left open, they are usually built to close automatically. They are arranged as follows (a) the door frame is set at an angle of 80°; the doors in these frames will slam of their own accord provided they are not opened more than 90°; (b) a piece of old rope is fitted to the door as a spring to keep it shut; (c) the door is provided with a counterweight to a rope passing over a pulley; (d) the hinges are set slightly skew. So that the door shall not be damaged by blows from mine cars or electric locomotives, buffers are built on them in the form of timber baulks or a spring bent piece of steel strip

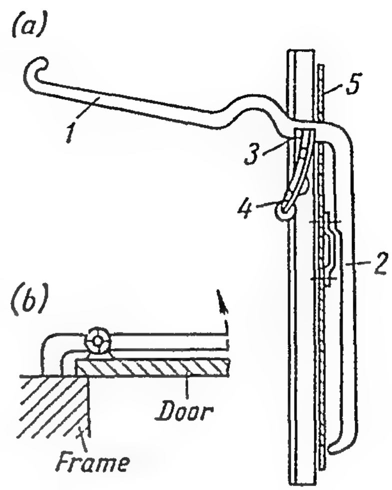

Fig 16-15 Levers to ease the opening of air doors

Fire steel doors are usually framed of angle iron or steel strip, with a sheet steel 3 mm thick fixed to them and stiffened with angles.

Doors of this type, heated by a fire, will buckle and cease to be airtight; more reliable types are so-called fireproof timber doors, covered with steel roofing sheet (3.5-4 kg/m^2) over felt or asbestos. Fire doors should not be built in roadways which are supported with timber

Automatic doors. In places where there is much haulage traffic, special men called trappers open and close the doors for the trains of mine cars. To avoid the use of trappers, doors are sometimes made to open automatically Many devices have been designed, including the following.

1. Doors which open by various mechanical drives actuated by the electric locomotive or mine car:

(a) when the wheel or its flange passes over a lever connected by a flexible or rigid link to the door;

(b) when the lever is rotated by the axle of the locomotive or mine car;

(c) ditto, operated by the body of the mine car or the frame of the locomotive,

(d) upon the movement of special ties, fixed to the door leaves

2 Doors opening by compressed air; by this means either compressed air passing into a cylinder will open the door through a connection between the piston and the door, or conversely the

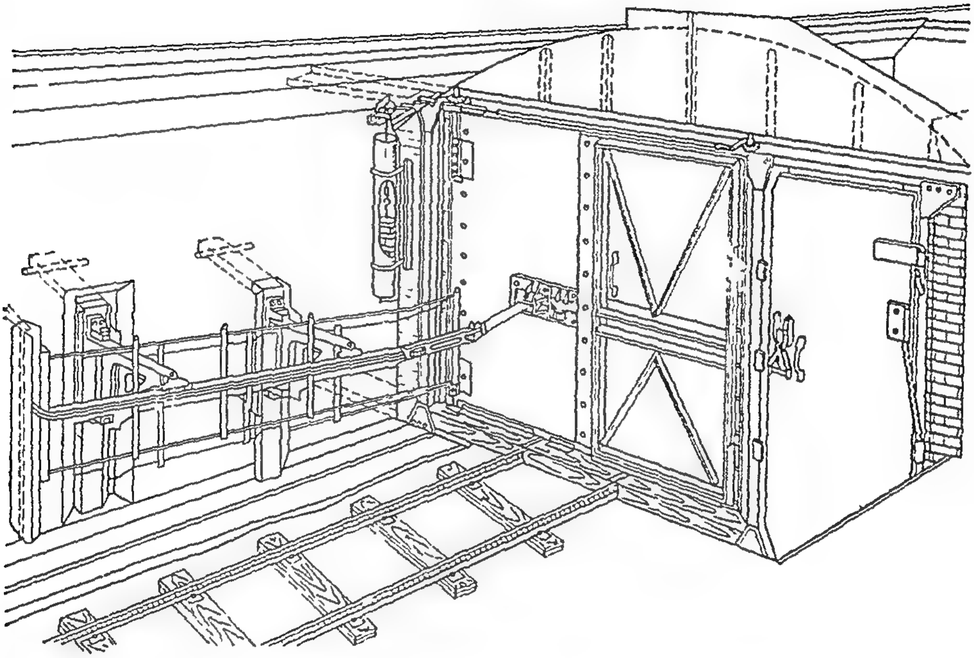

Fig 16-16 Automatic air door

door opens when air passes out of the cylinder. Entry or release of the air is controlled manually or automatically by a valve on the air pipe

3 Doors which open and close by a small electric motor automatically started by the train of mine cars

Some types of automatic door are shown in Figs 16-16, 16-17 and 16-18 Notwithstanding the fact that some 30 different types of automatic doors have been designed, none of them has been widely used The reasons for this are on the one hand, the arduous conditions for doors underground in areas subject to rock pressure, the dampness and dustiness of the air, and, on the other hand, neglect, particularly by the drivers of electric locomotives

The door in Fig. 16-16 was tested at mines in the Donets Basin and found satisfactory, when a mine car or electric locomotive

approaches it, the vehicle body presses the hinged lever which moves to the left and forward; the lever is linked to both leaves of the door, which open in opposite directions (eliminating the effect of a large pressure difference across the door); after the mine car has passed, the door closes under the effect of the counterweight; the joint between the two leaves is made tight with a rubber lining.

Figure 16-17 shows a door which works well in British mines; the door has a single leaf and swings upwards to open, as the mine cars approach the isolated section of the rail track a relay operates, actuating a contactor, which gives an impulse to start the drive of an electro-hydraulic pusher; to ease the action a small shutter is first opened (Fig 16-17).

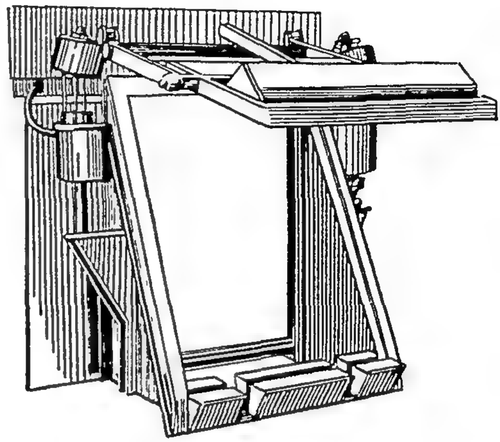

Fig 16-17 Single-leaf automatic air door

A simple folding door is shown in Fig. 16-18; on both sides of the door are tie bars 6 metres long, hinged to the door; when pressure comes on to the tie, the door folds and moves into the side, the door closes with a spring which does not need to be very strong.

9. **Trapdoors.** Also used for separating one air stream from another are trapdoors with one or two leaves, hung on door hinges.

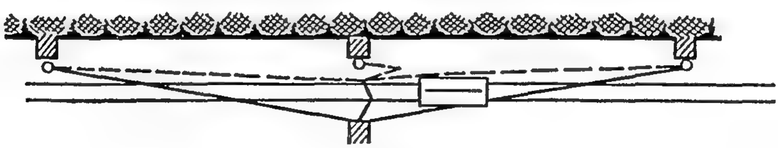

Fig 16-18 Diagram of folding air door

They are also used for closing off air from vertical or inclined shafts, blind pits, small pits, winzes, raises, etc.

Trapdoors are made of timber, hanging on a frame which may be of timber or 5 mm steel stiffened by angle irons and set in a concrete frame. They are usually built with counterweights to make them easier to open A grid of old rails is placed across the underside to prevent men falling through the open trap The trapdoor is placed at the top of the raise, at the bottom a brattice sheet is usually hung In particularly important places such as at the top of a small ventilating pit, trapdoors are installed in pairs to form an air lock,

10. Brattice Sheeting. In places where the air flow must be limited but not completely cut off, and where there is also considerable traffic of men and mineral, instead of a door a *brattice sheet* can be hung, consisting of a length of tarred canvas or other textile, nailed to a bar and freely hanging from it across the roadway Sheets are also hung in pairs One reason for using sheets in preference to doors is that heavy rock pressure puts doors out of action

11. Air Crossings. Air crossings are devices for enabling one airway to cross another

Air crossings should be strong, have a minimum aerodynamic resistance, and be airtight, preventing leakages from one stream to the other

The strength requirements are dictated by the fact that if the air crossing is destroyed by a gas or dust explosion, the resulting short circuit across it will deprive a whole district of intake air. In fiery mines air crossings must be of the strongest type.

Air crossings are built of: (a) timber *; (b) steel tubes; (c) concrete, reinforced concrete, or brick; or (d) a special by-pass road is sometimes excavated to make an air crossing.

Timber air crossings made of steel ventilation ducting are permitted for secondary airways in metal mines and in coal mines, respectively. For main airways, air crossings must be built of rivetted or welded tubes 5-7 mm thick and not less than 600 mm in diameter; if necessary, when there are no tubes of larger diameter, two or three tubes may be placed together; the tubes are 7-8 m long. Air crossings must be built so that men can pass through them in either direction, passage from one airway to the other without disturbing the ventilation takes place through air locks with double doors. To ensure strength and airtightness, the stoppings built at air crossings must be of round timber or masonry.

The wall separating one air stream from the other is built of one or two rows of rails or steel or reinforced concrete beams The resistance of air crossings is as follows when one tube of 0.6 m diameter (0 75, or 0 90 m) is used, it is 750 murgs (250 or 75); with two tubes of 0 6 m diameter (0 75 or 0 90 m) it is 500 murgs (150 or 50); a cross section of $0\ 75 \times 1\ 5$ m^2 has 20 murgs and a cross section of $2\ 5 \times 2\ 5$ m^2 has 10 murgs Air crossings are built with converging inlets and diverging outlets

The cross section of air crossings in secondary airways must be not less than 0 75 m^2, in main airways and when air crossings are built as by-passes they must be of 1.5 m^2 cross section. Sometimes

* Since they are not very reliable, timber crossings are only allowed during the initial development period of the mine.

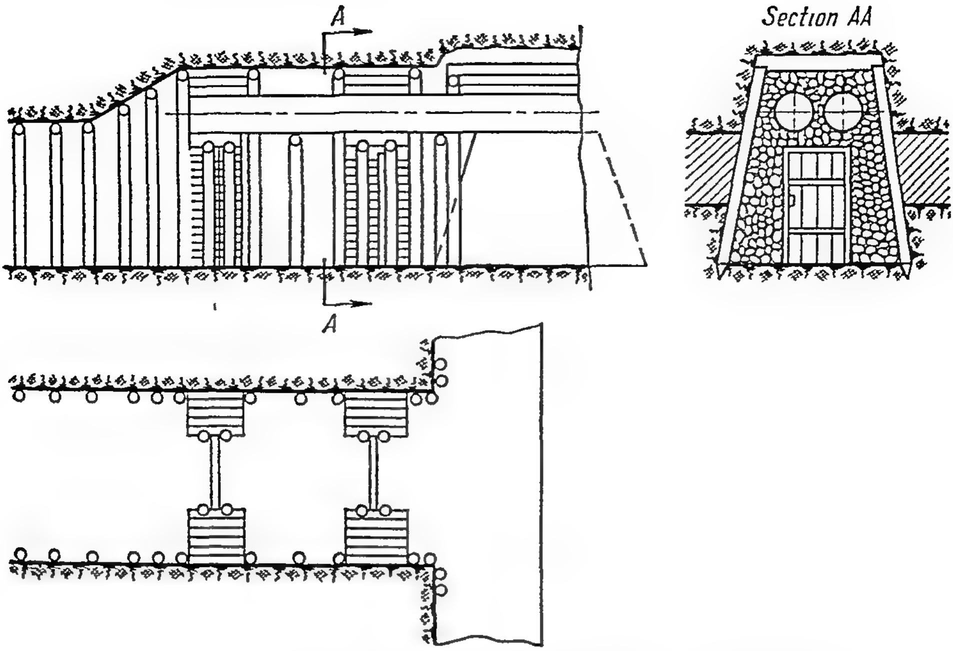

Fig 16-19. Air crossing made of round timber, and steel tubes

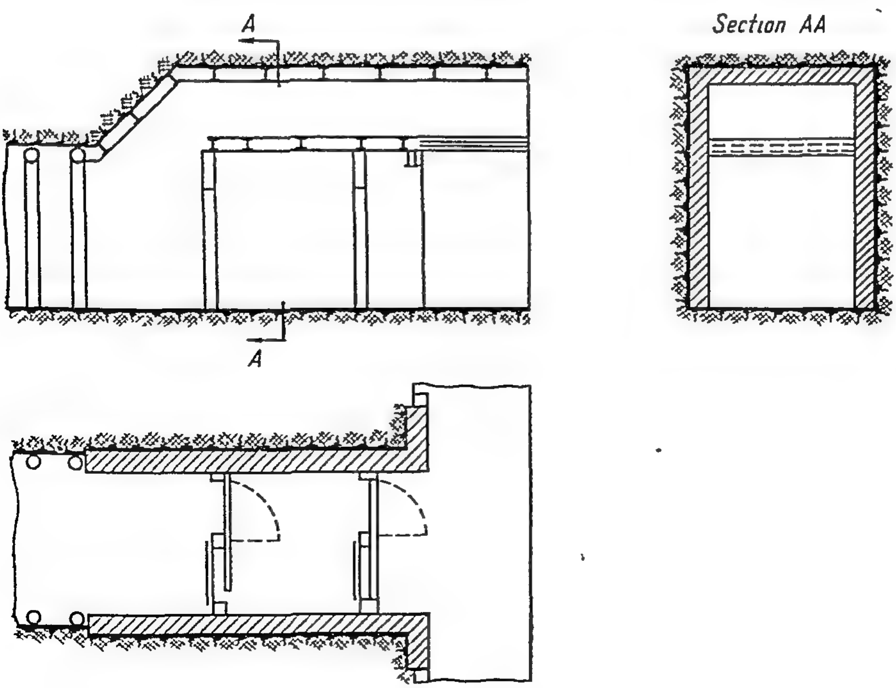

Fig. 16-20 Concrete air crossing of rectangular cross-section

the cross section of the air crossing differs little from that of the rest of the roadway. The pressure lost in an air crossing should not exceed 10-15 mm, and the air velocity according to the Safety Regulations should not exceed 10 m/sec. Standards of normal air leakage are stated in Appendix 5.

In well built air crossings using steel tubes the intake and return airways should be gradually narrowed and widened to reduce the

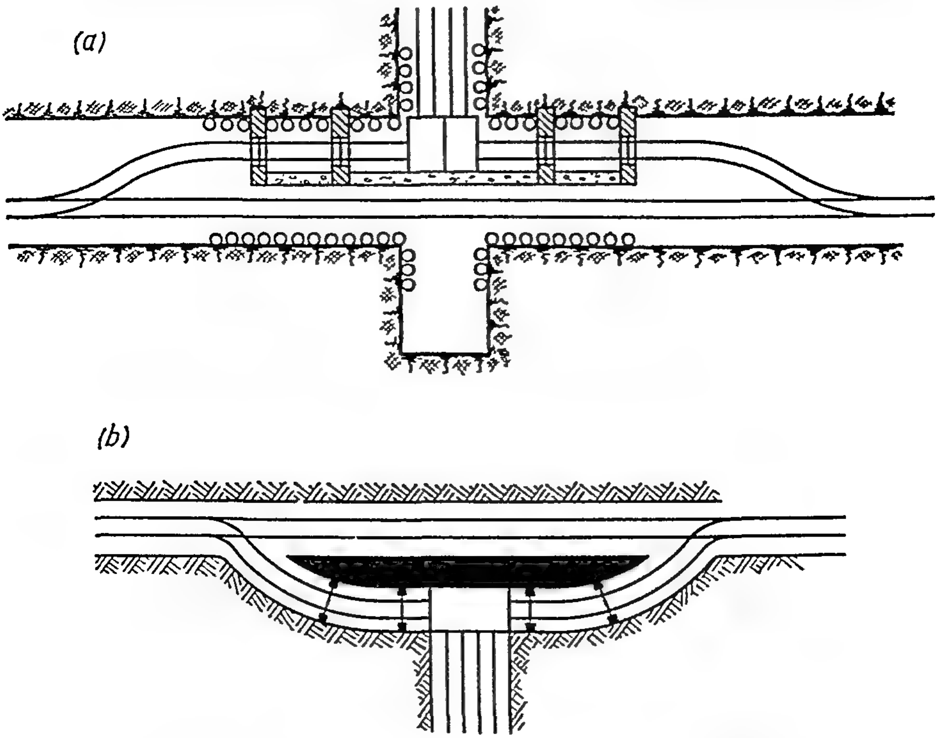

Fig 16-21 Layouts for the air lock at the foot of a haulage incline

resistance when the air is entering the crossing or leaving it With this end in view, air crossings should have simple converging inlets and diverging outlets or boards should be placed at a slope, or rock should be piled on both sides in front of the air lock doors, leaving the middle of the roadway free for men and mineral to pass, it is particularly important to aim at a smooth junction for the air crossing where the air stream widens into the return airway.

Figure 16-19 shows a timber air crossing with two tubes and Fig. 16-20 shows a concrete one

An air crossing built in the form of a by-pass road driven below or above the seam is preferable because of its low aerodynamic resistance; an air lock can be built in it

12. By-passes at the Foot of Inclines. By-passes at the foot of inclines prevent the intake air in the haulage road from passing up into the incline

These by-passes are built as follows·

(a) in the form of a wall (Fig. 16-21*a*) made of one or two rows of props faced with boards on one or both sides, plastered with clay or plaster and with the space between the two layers of boards rammed full of clay.

Stronger walls are built of concrete or of brick; on both sides of the by-pass there are two doors which open in different directions and form an air lock for men and mine cars to pass,

(b) as a coal pillar (Fig 16-21*b*); the incline is connected to the haulage road by a diagonal drift containing the air lock,

(c) as a rock pillar; the by-pass is built in the rock at the bottom of the incline, leaving a rock pillar some 7-10 m wide

By-passes of the last two types are more airtight than those built in the form of a wall.

13. Line Brattices. Line brattices are often used for separating intake air from return air when the flow passes only a short distance along one roadway. Since brattices usually collapse during an explosion, their use as permanent structures is allowed only in non-gassy anthracite mines or metal mines They are made of timber, brick or concrete, plastered on both sides With a large pressure difference across the brattice there will be heavy leakages

14. Ventilation Ducts. For the same purposes as line brattices, metal ventilation ducts of large diameter (1-1 2 m) are sometimes used.

CHAPTER 17

CONTROL OF VENTILATION. PRESSURE SURVEYS

17-1. GENERAL

Mine ventilation requires constant and systematic control. The resistance of the mine to ventilation is continuously changing because of the inevitable changes in length and reductions in the cross section of airways, the ventilation layout also changes The fan output therefore does not remain constant. On the other hand, there are variations in the quantities both of airborne dust and of the various gases emitted or formed underground which require dilution by the air. Finally, the natural ventilating pressure also changes in magnitude and sometimes in direction, while the temperature and humidity of the outside air coming underground also change. Because of these variations the composition of mine air never remains constant and needs control from the viewpoints both of safety and of health.

This control takes the form of

(1) measurements of air velocity and calculations of the quantities coming into the mine and individual sections and reaching the faces,

(2) sampling of the air and determining the content of various gases and dust in it;

(3) the measurement of air temperature, in deep mines.

Besides, pressure surveys are made to control the distribution of the pressure.

17-2. CONTROL OF THE VELOCITY AND QUANTITY OF AIR ENTERING THE MINE

The air velocity must satisfy certain minimum and maximum standards (Chapter 15, page 405 and Part Three, page 525)

Control of the air quantity provides the following.

(a) a knowledge of the total air flow passing down the mine;

(b) a knowledge of the air quantities supplied to and reaching various seams, sections, and faces;

(c) a check that the regulations are fulfilled insofar as they concern the requisite quantity of air per man underground and per ton of daily output;

(d) determinations of the air leakages.

It is desirable also to determine what quantity of air passes into the mine (Q_{in}) and out of it (Q_{out}) Generally, Q_{out} is larger than Q_{in} because of the heating of the air and its lower pressure under the rarefying action of the exhaust fan, and also because of the consumption of compressed air underground, of which large quantities may be used for driving engines

Continuous control of the mine air flow is provided by a recording flow meter.

Measurements of air velocity for determining air quantity must be made in measuring stations Measuring stations are located in straight lengths of airway, directly upstream and downstream of them there must be neither bends nor any structure such as a regulator which can redistribute the air flow across the airway; rock on the floor must be cleaned away; bellmouths (diffusers) at the beginning and the end of the station should be in close contact with the wall rocks and built so that all the air passes through the measuring station The station should be marked with a notice board stating the number of the station and the cross-sectional measurements at the point where the air velocity is measured; on this board there should be a space left for recording the quantity of air passing through the airway (in m^3/sec or m^3/min), the date of the measurement and the signature of the operator.

Measuring stations should be located at points where the following air quantities can be measured. the total downcast and upcast air, the intake air and return air for every seam or group of seams, as well as in the faces and in individual ore bodies Measuring stations must therefore be located near the pit bottoms in the haulage and return horizons, in ventilation inclines before the fan drift, in crosscuts, at the beginning and the end of haulage and return airways (in long haulage roads sometimes intermediate measuring stations are located every 500-1,000 m) Additional measuring stations are located wherever the air quantity must be measured for any reason.

If the face is a considerable distance from the last measuring station and the haulage road is encumbered, it may be necessary to measure the air as near as possible to the face; a place of measurement should be carefully chosen, attention being paid to whether the supports are close to the wall rocks or any gaps between the supports and the sides, particularly the roof, are carefully filled

Measurements of the air quantity should be made periodically; their frequency is stated in the Safety Regulations The measurements are recorded in a special book and transferred to the ventilation plan (Chapter 18). All changes in the location of ventilation structures and the direction of air flow must be noted by the ventilation engineer on the plan not later than the following day.

An air survey of the whole mine should be made in a non-working shift So far as possible all the air quantities should be reduced to the same specific weight (without which it is difficult to draw up a correct air balance), and simultaneously with the air velocity, the barometric pressure and the temperature must be measured. After the results have been reduced, if the work has been done correctly, the total air quantity should be equal to the flows in the separate splits

The air quantities are reduced to the general p_0 and T_0 by the equation

$$Q_{red} = Q \frac{pT_0}{p_0T}$$

where Q_{red} = air quantity reduced to an arbitrarily chosen pressure p_0 and temperature T_0

Q = measured air quantity at pressure p and temperature T in degrees absolute.

Usually even after the reduction there is a discrepancy between the quantities of air flow in the splits and the total quantity passing up or down the shaft; this is explained by

(a) variation in the ventilating conditions during the measurements, caused by the accidental opening of doors, the stopping of a booster fan, etc ;

(b) leakages at the surface or underground from one split to another, which are unaccounted for,

(c) errors in measurement

The simplest way of balancing the air quantities is as follows: let the measured quantities of air be Q_{total}, Q_1, Q_2, . . Q_n These air quantities are reduced first to the same p_0 and T_0 and the difference ΔQ between Q_{total} and their sum

$$Q'_1 + Q'_2 + \ldots = \Sigma Q'_n$$

is determined, where Q'_1, Q'_2, . . , Q'_n are the reduced air quantities; then the corrections to the individual values of air quantity will be

$$\delta Q_1 = \Delta Q \frac{Q'_1}{\Sigma Q'_n}$$

$$\delta Q_2 = \Delta Q \frac{Q'_2}{\Sigma Q'_n}, \text{ etc}$$

Example. The air quantities Q at a pressure p and a temperature t, °C are given in Table 17-1

In the sixth column of Table 17-1 are recorded the values of Q reduced to p_0 = 760 mm and t = 15°C or T_0 = 288°, the true air quantity is assumed to be the arithmetic mean of the first and fifth measurements, considered

TABLE 17-1. **Measurement of Air Quantity at Pressure p, and Temperature t, °C**

	Measurement point	Q, m³/min	p, mm	t, °C	Q_{red}, m³/min	δQ	Q_{cor}, m³/min
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
1	Pit bottom	1,500	780	5	1,595	+13	1,608
2	Split 1	400	781	6	424	—11	413
3	Split 2	500	770	10	516	—14	502
4	Split 3	700	765	12	712	—19	693
5	Main return	1,630	755	15	1,620	—13	1,607

as the most reliable $\frac{1,595+1,620}{2} = 1,608$ m³/min (if one of these measurements were more dependable than the other, it would be taken to be the most probable quantity of air)

The sum $Q_1' + Q_2' + Q_3'$ is equal to $424 + 516 + 712 = 1,652$ m³/min, $\Delta Q = 1,608 - 1,652 = -44$ m³/min The correction $\delta Q_1 = -44 \times \frac{424}{1,652} = -11$ m³/min, etc; consequently the corrected air quantity of the first split $Q_{cor} = 424 - 11 = 413$ m³/min By comparison of columns 3 and 6 it can be seen that mere reduction to the general specific weight considerably reduces the divergence in the measurements

If there are many measuring stations, and the person doing the measurements cannot complete them in one day, he must on the second day repeat the nth air measurement, Q_n, in the last measuring station, after which all the measurements of the first day are reduced to the values of the second day by multiplying by the ratio $\frac{Q_n'}{Q_n}$, or alternatively the measurements of the second day are reduced to those of the first day, using the factor $\frac{Q_n}{Q_n'}$

In the analysis of the air survey results, the following should be remembered since the cross-sectional area of the face varies very widely with the work being done in it, the resistance also changes, and consequently the air quantity flowing through the face also varies widely Because of this, the results of air surveys in the same mine can differ from each other even when the surveys are made within a short time interval.

The air flow in working mines is controlled by the Safety Regulations concerning limiting gas contents, and air requirement per ton of daily output At the same time in "Basic Indications for the Supervision of the Ventilation of Mines and the Fiery Condition of Coal Mines" confirmed by the Soviet Mines Inspectorate on August 1, 1957, it is stated that "the air flow needed for the normal ventilation of a district so as to dilute the gas shall be cal-

culated as follows

$$Q = 0.0764qA = \frac{qA\ 100}{24\ 60\ 0.9}\ \text{m}^3/\text{min} \qquad (17\text{-}1)$$

where q = actual or expected gas emission in the district, m³/ton of daily output

A = planned or actual average daily output of the district, tons

For districts or seams of gas Categories 1, 2 or 3, the value of the gas emission q is respectively equal to 5, 10, or 15 m³/ton of daily output"

The quantity of air given by Equation (17-1) is somewhat less than the quantity per ton of daily output calculated according to the Soviet Safety Regulations, i.e. for a mine of Category 1 it is divided by a factor of 2.6, for a mine of Category 2 by a factor of 1.6, and for a mine of Category 3 by a factor of 1 3. Thus, in working mines according to the Mines Inspectorate it is possible to give the district less air than according to the Safety Regulations, per ton of daily output but subject to the observance of the standards for the limiting methane content

In the "Basic Indications" there is no calculation for the air quantity needed for ventilating working mines, Equation (17-1) is unsuitable for this because it does not ensure for most mines (particularly very gassy mines) that the methane content in the general return shall be 0.75% or less; nor does it take account of air leakages For the observance of the CH_4 content (1 and 0 75%) and the corresponding air quantities in districts, calculated from Equation (17-1), it is doubtful whether the requirement per ton of daily output would be insisted on For urgent cases of gas emission, appropriate measures should be taken (see Part One).

17-3 CONTROL OF THE COMPOSITION OF THE MINE AIR

The composition of the mine air is checked according to the Safety Regulations by air sampling at the intervals stated, the samples being analysed in a mine rescue laboratory The results of the analyses are recorded in the ventilation log book. In addition, the content of the various gases is determined by portable gas analysers (Part One) and the dustiness of the air is determined in mine laboratories from samples taken underground.

17-4. CONTROL OF THE PRESSURE. PRESSURE SURVEYS

For control of the mine pressure, the mine main fan is provided with water gauges and their readings are recorded by the fan op-

erator in the log book. In all gassy mines it is mandatory to install recording water gauges (page 206). From the readings of the water gauge it is possible in some measure to judge the change in the total air quantity passing through the mine; if for a measured quantity Q_1 the water gauge reads h_1, then with a reading h_2 the air quantity will be approximately $Q_1\sqrt{\frac{h_1}{h_2}}$, i e. if h_2 is smaller than h_1, then Q_2 will be larger than Q_1 This conclusion is reliable when the fan is working on the right-hand downward arm of its characteristic, i.e. for most fans in practice. When the fan is working on the left-hand branch, i.e. in a mine with a very small equivalent orifice, the relationship will be reversed, i e *with reduction in the pressure, there will be a reduction in the air quantity*

The water gauge connected to the fan drift measures the mine pressure, the pressure of the fan drift (and of the bend from the shaft into the drift) up to the point of installation of the Pitot tubes connected to the water gauges, and the negative velocity pressure (Chapter 7)

The measurement of the total mine head does not solve the problem of its distribution throughout the mine But, for ventilating a mine and for proper control of the condition of the airways from the point of view of their resistance to the air passing through them, and to locate the "bottleneck" in the mine which make the mine ventilation difficult, etc , the distribution of the head is very important

The work of determining how the total mine head is distributed throughout the mine (including the natural ventilating pressure, if there is one) is called a pressure survey

The heads in the different mine airways can be measured

(1) by barometers, manometers, and other portable instruments; the head of any airway is equal to the difference between the readings of these instruments at the first and final stations of this airway;

(2) by using a micromanometer and rubber tubing, the head in the airway is read by a micromanometer to which rubber tubes are fitted, passing along the airway to its first and final sections.

The second method is cumbersome and requires much time and labour in setting and re-setting the rubber tubes; the survey in one mine can last from 10 to 15 days, an interval during which the ventilating parameters of the mine can change. Therefore below we will give a more detailed description of a pressure survey according to the first method, although the second method is fairly widely used in mines of the Kuznets Basin.

17-4 1 Instruments Used for Pressure Surveys

The instruments now used for pressure surveys can be divided into two groups:

(1) instruments of the barometer type, i e. consisting basically of a sealed box, in which the air expands or contracts, the resulting increase or reduction in volume of the box being indicated by a pointer,

(2) other instruments

The usual aneroid barometer belongs to the first group.

The special feature of the barometer, making it necessary to place it in a group by itself, is the delay that occurs when there is a rapid change in pressure, due to "creep". Even highly sensitive barometers are not free of creep; it has given an impression of unreliability to mine pressure surveys made by the barometer. When the barometer is carried along a horizontal or slightly inclined roadway, its readings, after correction and subject to the condition that the barometer was at the point of measurement not less than 20-30 minutes before the reading was taken, are reliable; even passing through air doors does not then greatly affect the accuracy of the readings. When passing up or down inclines and particularly up or down shafts, for some tens or hundreds of metres, the barometer readings are still lagging behind the true pressure even after 20-30 minutes, and only after passing through other airways do they gradually catch up with this pressure.* Thus, the pressure difference between two points on different horizons is understated, and the pressure of airways between these points (depending on the direction of the traverse) may be exaggerated Unfortunately, to give any indication of the magnitude of these errors is at present impossible because very little study has been made of creep in the mine

Aneroid barometers were described in Chapter 2

Of the instruments in the second group we will consider only the so-called "deprimometer" which seems to be the most accurate and suitable instrument for mine work

The first instrument of this type was proposed by D I Mendeleyev as early as 1875 and was then called a differential barometer Professor V B Komarov with A A Geskin developed the following simple construction of the instrument in 1937-1938

* Even lowering the barometer into the mine a day before the beginning of the survey does not eliminate this disadvantage of barometers, as shown by the work of the Mining Institute of Dnepropetrovsk (Professor F A Abramov)

A glass vessel *1* (Fig. 17-1) is lowered into a thermos flask, a space being left between the walls of the thermos and the glass vessel. The vessel *1* has a cock *5*; sealed into the vessel, but not reaching the

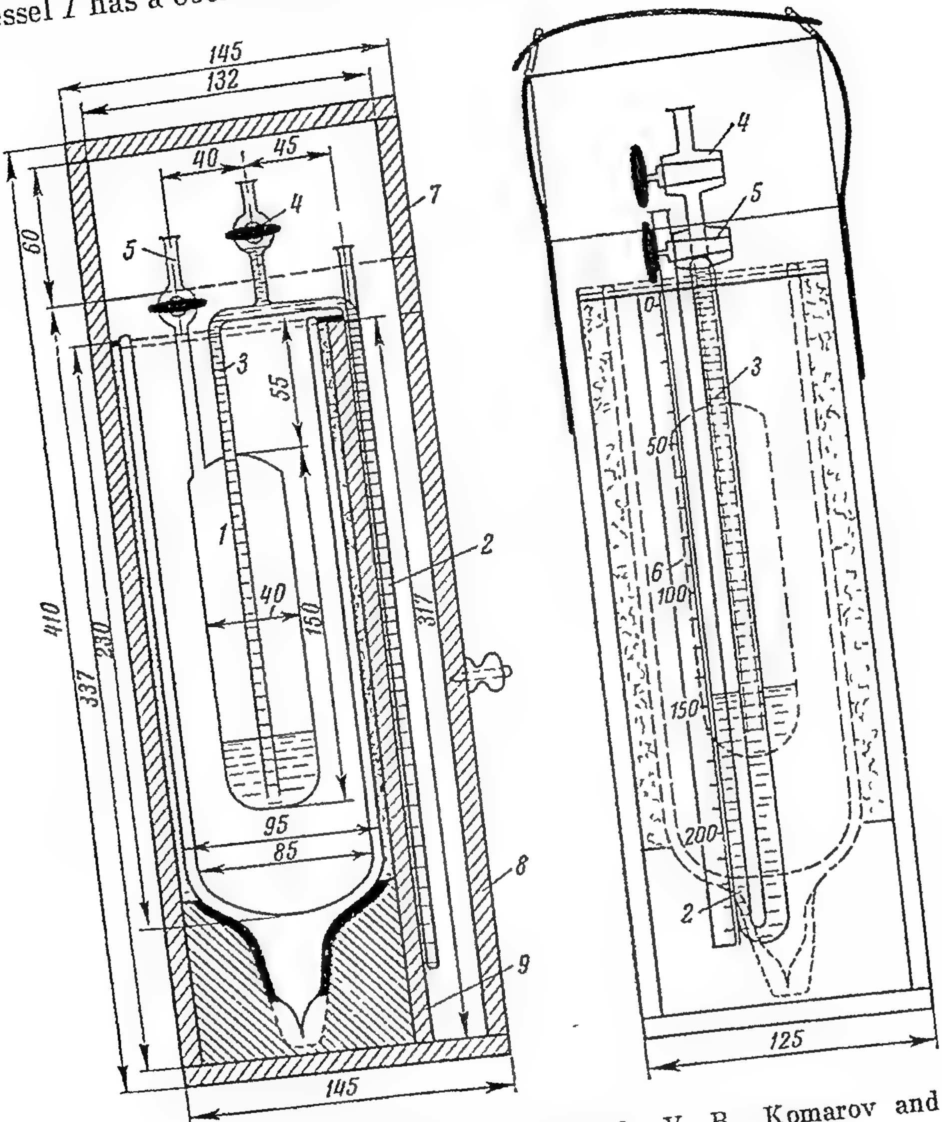

Fig. 17-1. Deprimometer, designed by V B. Komarov and A. A. Geskin

bottom is a tube *3* having a cock *4*, which is bent to form a U-tube *2* Along the left arm of the U-tube is a scale *6*. The thermos and the glass tubes are contained in a wooden box with a cover *7*, and a door *8* A partition *9* forms a compartment for the thermos and carries the tube *2* with its scale. The box measures 410 × 145 ×

× 125 mm *About 50 cm³ of kerosene is poured by a funnel through* the cock *5* into the vessel *1* with the cock *4* open; a short length of rubber tube is then attached to the end of the glass tube above the cock *4*, and the kerosene is carefully sucked up by the mouth until it fills the glass tubes and appears above the cock *4*, the cock *4* is then closed, with the cock *5* open, the liquid will stand in the open arm of the U-tube at the same level as in the vessel *1* The thermos is filled from above with lumps of ice

After the temperature in the vessel *1* has become steady, which requires 15-20 min, the cock *5* is closed.

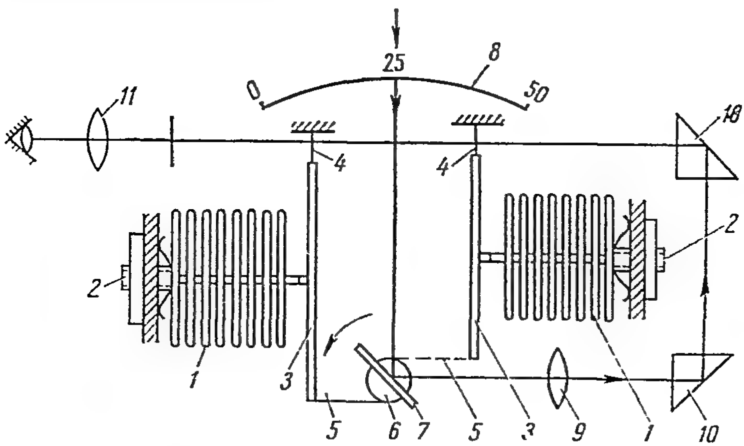

Fig 17-2 Diagram of the barometer-level of the Soviet Academician Shuleikin

The instrument works as follows Since the container is full of ice, the temperature of the air in it and consequently its pressure remain constant If the atmospheric pressure changes, the meniscus in the open arm of the U-tube will drop with increase of pressure, and rise with diminishing pressure. Consequently, if the original and final readings on the scale are noted, the difference between them, multiplied by some empirical factor, will be the pressure difference in mm water gauge. Change in the kerosene level in the vessel *1* can be neglected in view of the large difference between its diameter and that of the tube *2*

A disadvantage of the deprimometer is the need to fill the thermos with ice

These *instruments have been repeatedly used for* making pressure surveys and have been proved reliable, simple, and both accurate and convenient enough for underground use

Even more sensitive instruments than the deprimometer are available for measuring the absolute air pressure, for example the

"barometer-level" of Academician Shuleikin, a diagram of which is shown in Fig. 17-2. The elements which sense the change in barometric pressure are two aneroid units *1* connected by levers *3* with their ends joined by plates *4* to the instrument flange. The free ends of the levers carry the ends of sprocket chains *5* which are coiled on and anchored to the drum *6*. The chain tension is regulated by moving the aneroid units by means of the nuts *2*. When the barometric pressure changes, the units contract or expand, thus turning the drum *6* with the plane mirror *7* which is fixed to it The mirror reflects an image of the scale *8*, which passes through the object glass *9* and to right-angled prisms *10* to the eyepiece *11* with a stationary index. The index is a horizontal hair-line against which the readings are taken. The least scale division is usually from 0 07 to 0.11 mm of mercury. The eye can read half-divisions accurately, that is, to 0.5-0.75 mm water.

17-4.2 Procedure in Pressure Surveys

Pressure surveys consist in traversing the mine airways with one of the instruments mentioned above, while at each of the points previously marked, the pressure p_n of the air is measured as well as its temperature. If the measurements are made by the barometer at two points in the same horizontal plane, the difference between the instrument readings will be equal to the pressure difference in the length between these two points:
with the barometer,

$$h = (p_1 - p_2) \text{ mm water} \tag{17-2}$$

with the deprimometer,

$$h = k(D_1 - D_2) \text{ mm water} \tag{17-3}$$

where D_1 and D_2 are the pressure readings on the deprimometer.

The coefficient k is determined by comparing the instrument readings with those of a checked micromanometer; usually k varies between 0.95 and 1.05.

If there is a large difference in level between the beginning and the end of the length traversed, in Equation (17-2), which is based on the assumption of the incompressibility of air, there may be an error of up to 30%. The equation developed by B. I. Medvedev is more accurate for this instance

$$h = p_m\left(1 - \frac{p_2}{p_2'}\right) \tag{17-4}$$

where p_m = arithmetic mean of the air pressures at the start and the end of the length considered

p_2 = air pressure at the end of the length

p_2' = quantity calculated from the equation

$$\log p_2' = \log p_1 - 0.3443 \sum_{n=1}^{n=k} \frac{\Delta z_n}{R_n T_n} \tag{17-5}$$

in which p_1 = air pressure at the starting point of the length under consideration

Δz_n = excess elevation of the starting point over the end of the length, m

R_n = mean (for damp air) of the gas constant for this length

T_n = average air temperature.

If the measurements are made in an inclined or vertical shaft, then Equation (17-2) must include a correction for the pressure of the column of air between the first and second points Let the second point be H m below the first point (measured vertically) and the average specific weight of the air in the shaft be γ_{av}, then the pressure difference between the first and second points will be equal to

$$h = p_1 - \left(p_2 - \frac{\gamma_{av} H}{13\ 6}\right) \tag{17-6}$$

in the case when the second point is above the first

$$h = p_1 - \left(p_2 + \frac{\gamma_{av} H}{13\ 6}\right) \tag{17-7}$$

The average specific weight of air is taken as the arithmetic mean of the specific weights γ_1 and γ_2 at the first and second points,

$$\gamma_{av} = \frac{\gamma_1 + \gamma_2}{2}$$

The instrument readings show the effects of varying barometric pressure during the traverse

In measuring the pressure of one or two airways, these changes of barometric pressure will be so small that they are usually neglected, in a pressure survey of the whole mine or a large part of it, which may extend to 6-8 hours, changes in the barometric pressure in unsettled weather may be so considerable that neglecting them will lead to serious discrepancies in the final results of the survey the sum of the measured heads of all the airways in any ventilation circuit will not be equal to the total mine head measured at the fan To eliminate this error, throughout the duration of the pressure survey, readings of some control instru

ment such as a barometer are recorded, say at intervals of 15 minutes, the instrument being located at the surface or in the mine not too near the shaft so as to avoid the effect of the movement of skips or cages going up and down the shaft on the barometer readings

During the traverse of the airways the times of the readings are also recorded.

If the barometric pressure increases, then the correction in the form of a pressure difference from the readings of the control barometer will be added to the instrument readings, if the barometric pressure diminishes, the correction will be subtracted. Equations (17-6) and (17-7) will then take the form

$$h = p_1 - \left(p_2 \pm \frac{\gamma_{av} H}{13\ 6}\right) \pm (p_c'' - p_c') \qquad (17\text{-}8)$$

where p_c'' and p_c' are the readings of the control instrument at the moments when the two readings were taken on the traversing instrument.

The correction for differences in velocity pressures (Chapter 7, p. 236) is usually neglected

A traverse for a pressure survey of a whole mine requires two barometers * (one traversing and one control), one psychrometer (or thermometer), 2 clocks, one anemometer, one tape The team will include three persons two for carrying out the pressure survey and one for observing the control barometer

The main check on the correctness of the pressure survey is a comparison of the sum of the measured pressures Σh on the traverse (including the shaft) with the pressure of the mine h_m, measured in the way described on page 451, if the shafts are not included in the survey, their pressures must be obtained by calculation. Comparing the pressure h_m with the results of the pressure survey, it must be remembered that

(a) the natural ventilating pressure h_d is not included in the value h_m; when the pressure drop is measured during a pressure survey, the natural ventilation pressure is also measured (page 305);

(b) the natural ventilating pressure will differ along different traverses

According to this statement

$$\Sigma h = h_m \pm h_d$$

The value h_d can be determined by one of the methods indicated in Chapter 10.

The pressure survey should be made simultaneously with an air quantity survey; in this case with the air quantities passing

* If a deprimometer is used, only one control barometer will be needed and its reading will be compared at the start of work with that of the deprimometer

through the mine workings or sections, and with their pressures, it is possible to calculate their resistances and to work out, on the basis of these data, measures for improving the ventilation An electrical network calculator (see page 289) can be used for this purpose

According to Soviet Safety Regulations, mines which are difficult to ventilate must have a pressure survey every year.

Below are given some practical indications of how to carry out pressure surveys

(1) Pressure surveys (in particular those done with a barometer) can be made in mines with a total pressure not less than 20-25 mm.

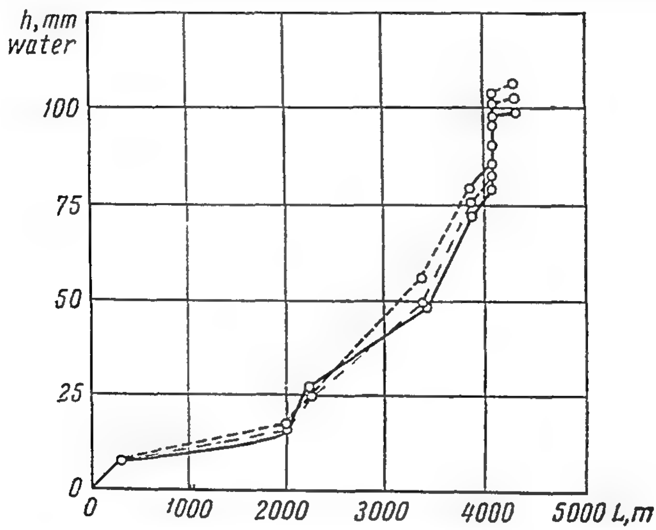

Fig 17-3. The results of a pressure survey after plotting

(2) Before the pressure survey is started, the route of the traverse should be established and the measurement points chosen in such a way that the pressure drop between them is of the order of 10-20 mm water, enabling the mine surveyor's office to give the elevations of these points The number of points should not be more than 15-20 so that they can be covered in seven hours

(3) The pressure survey should begin at a point near the pit bottom, at this point the cock *5* of the deprimometer should be closed (Fig 17-1)

(4) If the measuring stations are located on different horizons, all the points on one horizon should first be traversed, even though they belong to different air courses, and then the other horizon should be traversed

(5) If one or more points have to be passed several times, readings should be taken as a check every time they are passed.

(6) At each station, a sketch should be made of the airway, indicating the air measurement points, and the directions of air flow.

(7) If during the traverse, the meniscus in the tube *2* rises to the very top or falls to the very bottom, the cock *5* must be opened and a record should be made. "meniscus lowered (raised) by .. mm" and the cock *5* should then be closed

(8) At points of intersection of different airways, or when they are close to each other, the pressure difference between them should be measured, as a check

The results of a pressure survey can be plotted in a visually understandable way on a diagram with the abscissa axis carrying the spacing between the measuring stations, and the ordinate axis carrying the pressures at these stations (Fig. 17-3). *The steeper the curve rises for an airway, the less favourable is its ventilation from the point of view of resistance to air flow*

17-5. AUTOMATION AND REMOTE CONTROL OF VENTILATION INSTALLATIONS

The recent wide introduction of automation and remote control of machines in various industrial processes has also spread to mine ventilation.

Many varied devices have been proposed for automation and remote control of main fan installations and booster fans, and for the control of the pressure and the air flow.

Remote control of ventilation installations includes

(1) control of the air flow, the pressure of the fan, and of the temperature of its bearings, sometimes control of the attitude of the trapdoors and doors for reversing the air flow is also effected;

(2) control of the fan drive and reversing devices with a view to.

(a) remote starting and stopping of the fan motor,

(b) remote variation of the fan output by raising or lowering the door, or by turning the vanes of the guides in radial-flow fans;

(c) remote reversal of the air flow.

Automation of the control of the fan installation is understood to mean

(a) automatic change in the operation of the fan when the fan discharge changes because of changes in mine resistance;

(b) automatic stopping of the fan when it is faulty, automatic starting of a stand-by fan and simultaneous bringing into service of the appropriate doors or trapdoors.

The main requirements to equipment for automation and remote control are reliability of their operation and the operation of the

primary sensing instruments, which are subjected to fairly severe working conditions (for example, in the fan drift, in dusty and wet air), and simplicity of their circuits. Complete automation and control should not be aimed at where it is possible to do without them or where they might wear out the plant and impair the ventilation For example, there is no special need for remote control of the position of the trapdoors and doors since reversal of ventilation is very rare Full automation of the ventilating installation could lead to the following let us assume that because of the opening of a door, the fan output increases, because of the automatic reduction of the fan discharge and the short circuit, individual splits will be deprived of air. But when the resistance increases in a split, the total air flow reaching the mine is diminished, and the automatic control forces the fan to give more air, as a consequence of the changing resistances of the individual splits, the quantity of air in some of them increases

In these cases, it could be very useful to have remote control of the air flows in the splits, and remote control of the ventilation doors It must be remembered that short-term pulsations in the air velocity of the order of $\pm$25-50%, are always possible underground, and there is no need for automatic or remote control of this type of fluctuation of the air flow in any split The same applies to the regulation of the air flow with changing concentrations of harmful gases or airborne dust which at different points can vary very widely for relatively small changes in the mean contents of gas and dust.

Gipronikel (the State Institute for the Nickel Industry) has worked out the following automation schemes

(1) automatic (programmed) starting and stopping of the main and booster fans in metal mines, to arrange for intensified ventilation after blasting, and normal ventilation at other times

(2) automatic starting of the heating installation when the outside air temperature is below a certain limit

In the Kuznets Basin, the surface fans at subsidiary ventilating shafts have been automated, which is particularly valuable since there are a large number of such fans (15 or more) at some mines An even greater positive effect can be achieved by the remote control of these fans to enable the underground ventilating currents to be varied

Auxiliary fans can also be changed over to remote operation and interlocked with the equipment working in the face in such a way that when the fan stops, the power is cut off to all the equipment working in the district or mine It is also possible to arrange for operation of the auxiliary fans, for example, in the stopes in metal mines, so that the quantity of air delivered by them to the stope

depends on the degree of fouling of the mine air by harmful gas or dust.

The sensing instruments used, i e. the primary instruments whi respond to the change in air flow, are generally membranes of r berized textile, or panels moved or rotated by the air flow, a Pitot tubes combined, for example, with a ring balance.

The automation and remote control of ventilating installat improves their operational reliability and reduces the amoun staff required for the ventilation service. In case of emergen remote control enables the operator (acting under the chief eng of the mine) to establish the most appropriate ventilation c tions in the shortest possible time.

CHAPTER 18

THE VENTILATION SERVICE OF THE MINE

18-1. ORGANIZATION OF THE VENTILATION SERVICE

The ventilation service can be organized in different ways At some mines the head of the ventilation service is responsible for the ventilation economy as a whole, including the main fan and all booster fans, as well as for all the airways which are primarily used for ventilation, and for the lamp room; with this organization, the ventilation engineer will have under him a small mechanical repair staff and equipment In other mines the fans are the responsibility of the chief mechanic of the mine and of the chiefs of the underground districts. Sometimes the main fan is the responsibility of the chief mechanic alone.

The main members of the ventilation service include the ventilation engineer who is the chief, his assistant, and foremen; the auxiliary workers include operators for gas measurements (in gassy mines), stone-dusters, carpenters, door trappers, and mechanics for the auxiliary fans

In metal mines and in coal mines of a methane category not worse than No 2, the ventilation engineer may be of higher or medium mining technical education In coal mines of methane Category 3 or worse, only a mining engineer can be the ventilation engineer In mines subject to sudden outbursts, the ventilation engineer must also have not less than one year's service in gassy mines

The ventilation engineer is responsible to the chief engineer of the mine.

The ventilation foremen should be qualified miners, or men with not less than 3 years of underground experience, who have passed a special examination. Gas measurers are men with not less than one year's service who have passed a technical test.

The whole mine is subdivided into ventilation districts, served by the different ventilation foremen; in large mines these districts are usually the same as the producing sections; in small mines one ventilation district usually includes several producing sections; sometimes one or two ventilation foremen are solely responsible for the repair and construction work of the ventilation structures

18-2. MAIN WORK OF MINE VENTILATION SERVICE

The duties of the ventilation staff at the mine include *maintaining the mine air in a condition which is fully suitable for breathing, insofar as the contents of dust and explosive or poisonous gases and their permissible concentrations under the Safety Regulations are concerned, and also the provision of favourable climatic conditions for the miners.*

The fulfilment of these duties, according to the Safety Regulations requires·

(1) supervision of the condition of ventilation;

(2) observance of the gas and dust contents, and the action for dust suppression;

(3) drawing up and supplementing the ventilation plans. as well as keeping the ventilation books;

(4) the provision of an adequate air flow to all working faces by the regulation of the flows in the splits, and the reduction of air leakages,

(5) guidance in the repair of the airways, and of the ventilation structures which are significant for the whole mine,

(6) the provision of the ventilation equipment required by the mine.

18-2.1 Supervision of Underground Ventilation

Current Soviet Safety Regulations attach great importance to adequate frequency of supervision, and specify the following time intervals

(1) Air analyses and control of correct air distribution in coal mines are needed, once per month for non-gassy mines; for mines of Categories 1 and 2, twice per month, for Category 3 and above the categories as well as in the battery charging rooms of all coal mines, at least three times, and in metal mines at least twice per month.

(2) Gas measurements in mines of Categories 1 and 2 must be made at least twice per shift, in mines of Category 3 and worse, at least three times per shift

(3) Air analyses for the presence of NO_2 and CO in mines where shotfiring takes place in the coal and in the rock at least twice per month; in metal mines, at least once per month.

(4) The gas emission (to determine the category of the mine) is determined yearly.

(5) Analyses of the dust on the walls of the airways of dusty coal mines are made at least once every three months; the condition of

the stone-dust barriers is checked daily, the dust on the barriers is analysed at least every month; the dust is cleaned up at least once every month, and the airways are whitewashed not less than once every six months. In places of heavy dust formation, the dust is analysed daily

(6) The operation of the equipment for reversing the air flow is checked at least once a month.

(7) Main and booster fans are checked at least daily by the staff of the chief mechanic.

These fans are also personally checked weekly by the chief mechanic and the ventilation engineer

18-2 2 The Ventilation Plans

The ventilation engineer is obliged to draw up and to fill out monthly the ventilation plans. The *ventilation plan of a mine* is a plan of the underground work with the air flow indicated on it by arrows as well as the locations of the fans, whether main or booster and other ventilation structures such as air crossings and doors On the plan are recorded also brief information on the fans (type diameter, and rotational speed) and the results of the last air measurements, the locations of the stone-dust barriers and stores of stone dust are marked, also the refuge rooms, stores of emergency material for firefighting, fire stoppings and fire doors; the intakes and returns are marked with different colours (red and blue, for example)

The *ventilation layout* is a schematic diagram showing the locations of the fans and the direction of air flow in the main airways Airways are usually indicated by a single line on this diagram. Sometimes the diagram is made clearer by drawing it as an axonometric projection, this is helpful at mines which are working several seams

So that the information on the ventilation plan can be used for drawing up a ventilation layout, the ventilation plan should have index letters on it marking each place where the splits separate and re-unite, then, beginning from the collar of the downcast shaft, a layout is drawn up, in which the beginning and the end of each split should be shown by means of straight or broken, or curved lines and these points being indicated by the same letters as on the ventilation plan. The ventilation layout is at first drawn up roughly, and then it is re-drawn finally in such a way that it gives a good visual impression and shows the features of the ventilation of the mine The main advantage of this layout is that it gives a good clear-cut picture without loss of accuracy; the drawing is not to scale.

If the same drawing shows the air flows at different periods of operation of the mine, the air streams for the different periods must be indicated by pencil or ink washes of different colours.

18-2 3 Ventilation Record Books

The ventilation engineer is obliged to keep the following log books

(1) the ventilation log book in which the results of measurements and analyses of the air are recorded as well as reports on the results of reversal of ventilation;

(2) a corded book (for gassy mines) for recording the percentage methane contents,

(3) a corded book (in mines which work seams subject to sudden outbursts) for recording all incidents of gas emission and outbursts of coal, and a log book of shock blasting (inducer shotfiring);

(4) a corded book for recording the stone-dusting of airways, the condition of the stone-dust barriers, the analyses of the dust on the stone-dust barriers and the stone-dust taken into the mine;

(5) a working log book in the fan building for recording the water gauge readings, the times of operation of the fan, and the results of inspections of the fan installation.

18-3. PROVISION OF THE AIR FLOW TO THE WORKINGS. FINDING OUT CAUSES OF FAULTY VENTILATION

The provision of enough air to all the working places in the mine is the main duty of the ventilation service.

When the quantity of air passing into the workings, including the faces, is inadequate, the ventilation engineer must be able not only to find out the causes of faulty ventilation, but also indicate how to eliminate or at least reduce these deficiencies

An investigation into the condition of mine ventilation should be made in the following sequence:

(1) inspection of the operation of the fan installation,

(2) inspection of the condition of ventilation of the workings

The indexes characterizing the operation of the fan installation are.

(a) the percentage of air short-circuited at the surface (for normal levels see Chapter 14);

(b) the resistance of the fan installation R_{inst} (for standards see Chapter 11) equal to

$$R_{inst} = (h_f - h_m)/Q_f^2 \text{ kilomurgs} \qquad (18\text{-}1)$$

(c) the efficiency of the fan and its operating conditions, i.e the operating point on the characteristic; the efficiency should not

be less than 0.6 and the operating conditions of axial-flow fans should be on the right-hand downward branch of the characteristic (not higher than 0.9 h_{max});

(d) the efficiency of the fan installation

$$\eta_{inst} = \frac{h_m}{h_f} \tag{18-2}$$

showing what part of the total head developed by the fan is spent on ventilating the mine; with small fans it is equal to 0.6 to 0.7, with medium fans it is 0.65 to 0.80; with large fans it is 0.75 to 0.90,

(e) the total efficiency of the fan installation η_t, equal to the ratio of the power in the air coming out of the mine to the power on the fan shaft,

$$\eta_t = \frac{N_m}{N_f} = \frac{Q_m h_m}{102} \quad \frac{Q_f h_f}{102\eta_f} = \frac{Q_m h_m \eta_f}{Q_f h_f} = \eta_{inst}\eta_f \frac{Q_m}{Q_f} \tag{18-3}$$

The value of η_t is 0.5 at best, usually it does not exceed 0.4, but with large short circuits of air and a high resistance of the fan installation it will barely reach 0.2-0.3.

In the equations above

h_f = total fan head obtained from the overall characteristic

Q_f = fan discharge

Q_m and h_m = quantity of air coming out of the upcast shaft and the mine head.

The condition of mine ventilation is characterized by:

(a) the mine resistance or its equivalent orifice;

(b) the quantity of downcast air, and of intake air to the faces;

(c) the air leakages (for standards see Chapter 14)

An inspection of the fan installation and the condition of mine ventilation will show whether the cause for an inadequate supply of air to the face lies in the fan installation or in the mine.

The causes of unsatisfactory operation of the fan installation include:

(a) high resistance of the fan installation caused by a restricted fan drift with bends in it, freezing up (filling with ice), filling with dirt (dust), the presence of old timbers, the absence of an evase, etc.;

(b) high short circuits at the surface and through the fan drift for the stand-by fan, or through the reversing doors,

(c) low fan efficiency causing uneconomical running;

(d) location of the operating conditions of an axial-flow fan at the top of its characteristic, which may lead to unstable operation upon reduction of the equivalent orifice of the mine.

Methods of eliminating the deficiencies in the fan installation were described in Chapters 14 and 16; when the fan drift (or the fan

rotor) is frozen up, the fan drift must be heated (this refers mainly to axial-flow fans with exposed fan drifts) and the short circuits of cold air from outside must be reduced

In the mine the ventilation can be judged by.

(a) the results of an air survey of the mine;

(b) analyses of air samples,

(c) the mine ventilation plan,

(d) information obtained during a traverse of the mine airways and an inspection of the ventilation structures

The causes of poor ventilation are.

(1) A faulty ventilation layout, affecting the supply of air to the faces unnecessarily long airways, a large number of doors and air crossings; high pressure drops through scale doors; close location of return and intake airways, faulty locations for fan installations.

The length of a ventilation circuit can sometimes be reduced by driving an additional road or by directing the air along another course; instead of a scale door of high resistance, it is often possible to install a booster fan in a split receiving not enough air, undesirably close location of the return and intake airways can sometimes be eliminated by changing the layout, special attention must be paid to the location of fans; it is often possible to greatly improve the ventilation by re-locating existing fans, or installing new ones, or changing inefficiently operating fans

(2) Poor condition of the underground airways An inspection of the airways may show which of them require widening or repairing (very useful information may be given by the very simplest pressure survey, made with a barometer through some airways); to reduce the resistance of difficult airways it is sometimes helpful to face the timbering with smooth material or to change the supports for others with a lower coefficient of friction; sometimes it is possible to direct the air through two airways instead of one; air crossings of small area and with restricted approaches should be rebuilt.

(3) Large air losses. Methods of reducing air losses have been discussed in detail in Chapter 14 The locations of losses are found by comparing the air quantities in the various airways on the ventilation plan.

(4) Unsatisfactory ventilation of development workings as a consequence of incorrectly located fans, large air losses in ventilation ducts, inadequate power of auxiliary fans, etc

18-3.1 Distribution of the Air

The duties of the ventilation engineer include not only the supply of enough air to the mine but also its correct distribution in

the districts and working places. In each individual case the ventilation engineer should decide whether it is possible to obtain the required distribution of air with stoppings and regulators or whether it is necessary to install auxiliary fans

18-3.2 Repair of Airways

Management of the repair of airways and the reconstruction and maintenance of various ventilation structures (stoppings, doors, air crossings, by-passes at the foot of inclines, air locks, trapdoors, etc) are also included in the duties of the ventilation engineer and his assistant

18-3 3 Provision of the Necessary Ventilating Equipment and Materials for the Mine

The proper operation of the ventilation equipment and the reduction of the gas and dust contents of the air to the specified levels is included in the ventilation engineer's duties

The ventilation engineer also takes part in the following work:

(a) determining the methane category of the mine;
(b) checking the reversing of the air flow;
(c) drawing up a plan for mine recovery work.

Determining the gas category of the mine As pointed out above, this determination must be made once a year (in June and July) It is made from the maximum of three observations per month, each observation being made 3 times per day (every month) The quantity of gas emitted is calculated for the mine as a whole as well as for separate seams and wings

The quantity of gas emitted into the different splits per day is calculated from the equation

$$V = 60\,(Q_1 n_1 m_1 + Q_2 n_2 m_2 + Q_3 n_3 m_3)\ \text{m}^3 \tag{18-4}$$

where Q_1, Q_2, Q_3 = averages of 3 measurements of air quantity in the return airway in each of the 3 shifts, m³/min
m_1, m_2, m_3 = fractional gas contents (for example, 0 5% is expressed as 0.005)
n_1, n_2, n_3 = duration of each determination, equal to the interval between the two closest observations (7 hours).

For each value of V, for the given split, the gas quantity per ton of corresponding average daily output is calculated, T_{av} (per wing, per seam or per mine) for a month

$$q_{gas} = \frac{V}{T_{av}}\ \text{m}^3/\text{min} \tag{18-5}$$

The maximum value of q_{gas} is used for determining the gas category of the mine, the gas emission from the gassiest seam or wing being taken as the mine category.

Since the emission of CO_2 sometimes exceeds that of CH_4, the category of the mine must also be decided for CO_2.

If seams are of different categories, the problem arises which category should be assigned to the mine as a whole If ventilation for the removal of gas is concerned, the category of the mine should correspond to the gas emission from the gassiest seam Standards of air quantity are discussed in Part Three. Meanwhile we must point out the following when the mine output for any reason falls considerably below its normal level, there is a reduction in the quantity of gas (methane or carbon dioxide) but the latter does not reduce in proportion to the output and therefore the quotient of the gas quantity divided by the output in this period of lowered production appears to give a heightened gas yield per ton and the mine could be transferred to a higher category.

This explains why in some mines of Category 3 and higher, the gas content in the general return air sometimes does not exceed 0.1 to 0.2 per cent.

If several fans are in operation at the mine, then during the determination of the category it is necessary to make sure that the ventilating conditions do not change, i e. that during the measurements of air quantity, the air circuits are controlled by the same fans.

Reversing of the air flow. The ventilation engineer is responsible for the proper operation of the reversing equipment A simple reversal of the direction of operation of the air from exhaust to forcing (or vice versa) should be done, as pointed out, once a month; the time of reversal is fixed by a certificate and it should not exceed ten minutes It is considerably more complicated to reverse the ventilation and simultaneously to follow up the air distribution in the airways. The ventilation doors underground are usually forced onto their frames by the air pressure and when the direction of air flow is changed they open, altering the air distribution; to prevent this, double ventilation doors should be built, which open in opposite directions so that when one door is opened on reversal of the air flow, the second is closed; it is also possible to build doors with fasteners or, when the air flow is reversed, to place door trappers at them; only when this is done, is it possible to expect that reversal would give about the same air distribution as normally.

Reversal should be done in a holiday or on a repairing shift. When the air flow is reversed, the fan should give not less than 60% of its normal output, but in individual cases when the methane content in the general return air is less than 1%, it may be reduced below 60%.

If booster fans work at the mine and the plan for the recovery of the mine after an emergency provides for their use, they also should be provided with devices for reversal

When the air flow is reversed in gassy mines, it must be remembered that after reverting to the normal air flow, the same air will pass through the face at least twice and sometimes three times, and consequently its methane content will be higher than usual Bearing this in mind, it is necessary to take the following precautions when testing the air distribution the men must be removed, the electric power must be cut off, and normal ventilation must be continued for long enough after the reversal before the work begins again Tentatively it can be taken that after the checking, before allowing work to re-start and switching on the power, the fan should work normally for 15 minutes in non-gassy mines and mines of Category 1, for 30 minutes in mines of Category 2, for 45 minutes in mines of Category 3 and 1 hour in mines outside the categories, after which a check must be made that there are no accumulations of gas.

An investigation of reversal procedures in the mines of Donets Basin undertaken by N A Frolov of the Novocherkask Polytechnic Institute brought out the following conclusions

(1) the reversal of the air flow throughout the mine takes 2 or 3 minutes,

(2) the amount of air delivered by the fan after reversal is reduced by the effect of the natural draught, in the course of time, as the temperatures of the downcast and upcast air change, the natural ventilating pressure can diminish and then change its direction;

(3) observations in one mine on the change of the composition of the air after a second reversal showed that the increase in the gas content of the return airway was insignificant (this statement requires confirmation)

PART THREE

Planning of mine ventilation

A mine ventilation plan should be in harmony with the mining plan Frequently when the mine ventilation is planned, corrections have to be made to the project for opening up, developing and producing the mineral; the driving of additional airways has to be considered, the cross sections of some airways have to be increased, and the method of roof control may have to be changed.

The mine ventilation plan controls the air supply to the faces, and consequently the safety and health of the workers A poorly drawn-up and realized ventilation plan can greatly increase the cost of ventilation or involve new capital investment for its rearrangement.

III-1. THE CONTENT OF THE VENTILATION PLAN

The mine ventilation plan consists of the following sections

(1) the choice of the ventilation layout—the method of ventilation and the location of the fan installation;

(2) a calculation of the total air flow required by the mine;

(3) a calculation of the head created by the main fan;

(4) the choice of the fan (or fans) for the mine and of its (their) drive,

(5) the design of the heating installation;

(6) a calculation of the cost of ventilation

The ventilation must also be planned for the period of entry into the deposit and development of the mine.

III-2. DEFINITIONS

The *ventilation layout* is a diagram on which are shown the locations of the fans and the direction of air flow in the main airways. A ventilation layout is also understood to mean a particular arrangement of the main intake and return airways to the mine

Return airways are those which remove the stale air from the faces to the surface and they include the return airway from the face and any upcast shaft or small pit.

The *return horizon* is the totality of the return airways which are at about the same level

Methods of ventilation The following methods of ventilation exist

(1) natural ventilation caused by the natural ventilating pressure;
(2) artificial ventilation by a fan or other devices.

Artificial ventilation includes

(a) exhaust ventilation,
(b) forced ventilation,
(c) combined (forced and exhaust) ventilation

Note. Artificial ventilation is generally accompanied by natural draught

III-3. VENTILATION LAYOUTS

III-3 1 Drawing up the Ventilation Layout

The ventilation layout is drawn up during the ventilation planning usually in two stages first the location of the main mine fan and the method of ventilation are decided, then the directions of movement of the air within the section or around the stopes are settled

When the ventilation layout is drawn up, safety is the first consideration, together with reliability and economy of ventilation With this in view, before the drawing up of a layout is started, it must be decided whether the mine will be a single ventilating unit, or if it is large, will it be more convenient to divide it into two or more independent units or sections to the dip, to the rise, or both

The ventilation layout can be drawn up for the whole life of the mine or, in case it is too long, for 15-20 years of work. Sometimes a separate ventilation layout is drawn up for the mining of the upper horizons (the first one or two horizons) with a part of the mine take extracted by an adit or by a brake incline

All mines, as a rule, should have artificial ventilation Exceptions are made by agreement with the Mines Inspectorate for metal mines which are not subject to danger from gas, silicosis, self-heating, or explosion of sulphide dusts, provided that the air analysis corresponds to the requirements of the Safety Regulations

Main mine fans should be at the surface but, as an exception, may be installed underground, underground installation of booster fans requires special permission

In gassy mines, only exhaust ventilation is allowed, forcing ventilation is allowed for all metal mines, non-gassy coal mines and the topmost horizon of mines of every gas category, combined (forcing and exhaust) ventilation is authorized for all coal mines provided that the coal faces are under depression

If there are more than two outlets to the surface, it is desirable for them all to be in use either as intakes or as returns

The atmospheric air which passes down a vertical or inclined shaft should be directed to the lowest producing horizon, along the airways of this horizon, the air passes to the faces or stopes, sweeping the faces in a rising direction, after which it passes along the return horizon to the upcast shaft and out to the surface This is a generally accepted layout; it corresponds to the natural rising movement of the air heated in underground workings, with this arrangement all the gases and dust contained in the air are carried out to the return air horizon and do not foul the intake air

This layout has exceptions

(a) when the deposit is a horizontal one, naturally there is no rising flow to the faces;

(b) when a mine take is extracted by a brake incline, the Soviet Safety Regulations allow the return air flow in non-gassy mines to be directed downwards along a travelling way to an airway driven parallel with the haulage road; in gassy mines this authorization is limited to angles of dip below 10°, with this layout there is no need to sink a ventilation pit at the outcrop of the seam,

(c) in metal mines using top slicing, a downward direction of the return air is allowed when there is no return airway driven in the rock; it is also allowed in a retreating method of working when the return airway in the ore is abandoned; in these instances the layout should provide for a return airway located at the same horizon as the haulage road or slightly above it

Other exceptions will be considered below in the description of the ventilation layouts of mine sections

To reduce the pressure the air flow should be divided into separate streams with approximately equal pressures

If this rule is not observed, use has to be made of stoppings with regulators which in certain cases offer a high resistance and cause a useless consumption of power, for example, the cost of overcoming the resistance of one regulator with a pressure drop of 50 mm and with a quantity of 20 m^3/sec of air passing through it at a power cost of 10 kopeks per kWh would be about 15,000 roubles a year

Layouts which have a large number of regulating doors or air crossings must be avoided; they are unreliable and result in heavy leakages of air

To prevent heavy leakages, layouts in which air currents at large differences in pressure are too close to each other should also be rejected.

In mining the topmost and sometimes also the second horizon of a series of seams or of several separate ore bodies, a layout with one central forcing fan obviates the use of a large number of fans

It is forbidden to remove return air from working faces through falls or caved areas

In mines working dusty seams, it is forbidden to supply fresh air through the skip shafts or inclined shafts for belt conveying of coal

Below is given some additional information for coal mines and metal mines

Coal Mines 1. The layout should provide for a return air horizon; this horizon will not naturally exist in a horizontal seam, nor is there any need for a general ventilation horizon at any mine where each section or group of sections is ventilated by a separate fan (the Kuznets coalfield, Prokopyev area; the Estonian shale mines, etc).

2 As pointed out above (page 460), the direction of the return air current in gassy mines at any angle of dip above 10° should be ascensional, but with downward movement of the air, its velocity should be at least 1 m/sec, and the methane content not more than 0 5%. For faces shorter than 50 m, the angle of dip is unlimited

3 Series ventilation of several faces is permitted in the mining of seams which are subject neither to sudden outbursts nor to blowers of gas, if the total length of the faces being ventilated is not above 300 m, and some additional intake air is provided to each face from intermediate (gate) roads, reducing the methane content in the air coming from the lowest face to a concentration not above 0 5%

Metal Mines 1. A general return air horizon is also desirable at metal mines.

2 The layout chosen should ensure through ventilation of the faces, and this often involves rock drivages for the return airways and raises.

3 Individual stopes should be ventilated independently from each other; series ventilation is permitted for not more than two stopes provided that the first stope has an additional intake air supply or that the dust is separated from the air by water curtains or mist.

These are the most important requirements in the drawing up of a ventilation layout Sometimes to find the best solution it is necessary to make a cost comparison of two or three layouts Modifications may also have to be made to the method of entry into the deposit or to the method of mining.

III-3 2 Direction of the Air Flow

All the ventilation layouts considered below can be included in one of the following three groups according to the direction of air flow·

(a) central,
(b) boundary;
(c) combined.

With the central layout of ventilation the air circuit is U-shaped, that is, the air moves at first to the faces in one direction and then along the return airway passes back to the shaft located in the centre of the mine take. With the boundary layout the air flow is direct, that is, the return air does not pass back but moves out to an upcast shaft located at the boundary of the mine take. With the combined layout, both the U-shaped and the direct air flow are used

With the central ventilation layout, and an advancing method of mining, a return airway is maintained to take the air out to the upcast shaft. To do this either coal pillars of adequate dimensions must be left on both sides of the return airway (in coal mines) or, the return airway must be driven through the stone (in metal mines).

III-3 1 Location of the Hoisting Shaft and the Ventilation Shaft

Figure III-1 shows the main possible layouts, and the interrelation between the upcast and downcast shafts

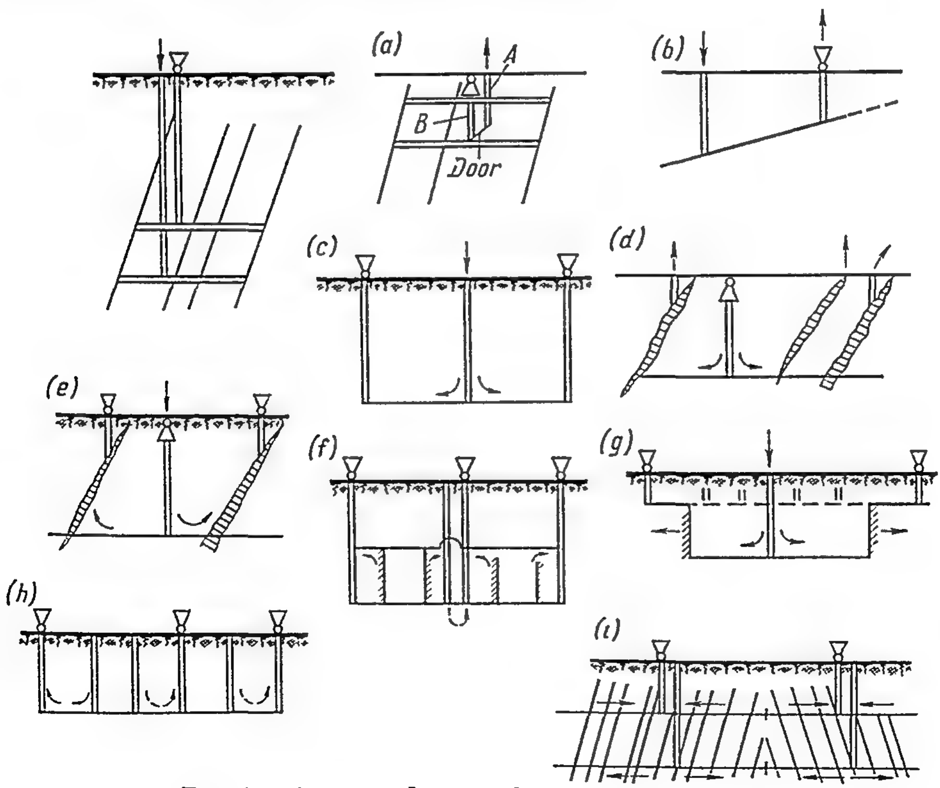

Fig III-1 Basic layouts for mine shafts

(a) both shafts—hoisting and ventilation—are placed together approximately in the middle of the mine take—a layout which can be described as a *two-central-shaft* layout (Fig. III-1*a*);

(b) the ventilation shaft is located on the same dip line as the hoisting shaft, but above it to the rise, and one or more horizons higher—the so-called *central* location (Fig III-1*b*),

(c) the fans are located at the boundary of the mine take at inclines or small pits; this is the *boundary* location (Fig III-1*c*); depending on whether the fan at the boundary serves a whole wing of the mine or several sections or only one, three layouts are distinguished boundary wing layout, boundary group layout, boundary section layout,

(d) a forcing fan is located at a central shaft, and the air passes out through small pits at the boundaries (Fig III-1*d*);

(e) in addition to the central forcing fan, boundary exhaust fans are installed (Fig III-1*e*);

(f) in addition to the central exhaust fan, boundary exhaust fans are installed (Fig III-1*f*);

(g) to ventilate the topmost horizon, every 100 to 300 metres, depending on the depth of the deposit, small ventilation pits are sunk and as the work advances fans are transferred (Fig. III-1*g*);

(h) for shallow deposits, the take is divided into a series of comparatively small independent sections, each served by two small pits—a downcast and a return (Fig III-1*h*),

(i) in large mines the mine take is divided into two to three independent ventilation units (Fig III-1*i*),

(j) when the method of mining is through adits, a forcing fan is placed at the mouth of an adit or an exhaust fan is placed above an upcast pit.

Possibilities and Combinations of Different Variations. The layouts shown in Fig III-1*a* and *b* are central layouts, the layout of Fig. III-1*f* is a combined central-boundary layout, all the other layouts in Fig III-1 (except *i*) are boundary layouts.

The advantages and disadvantages of the central layout of ventilation are stated in Table III-1

The power consumption, as pointed out by V V Vladimirsky, is the same for central and boundary ventilation layouts; however, taking into consideration the fact that with the boundary layout the fan operating conditions are more or less constant, and consequently it is possible to choose a fan to work at a high efficiency, the power consumption with the boundary layout will be rather lower than with the fans centrally located The lowest cost is possible with the fan shaft located at the middle of the wing of the take

The two-central-shafts layout is used most often with relatively deep deposits (more than 150-200 m) which have no outcrop at the earth's surface, the upper horizons of an outcropping deposit are generally ventilated by small pits, but when they have been mined out, the ventilation changes over to the use of two central shafts

TABLE III-1 **Advantages and Disadvantages of Ventilation by Two Central Shafts**

Advantages	Disadvantages
Faster start to production	Larger air losses at the lower horizon through the worked-out areas, and hence greater danger of fire in mineral subject to self-heating
Smaller capital outlay before production starts	
Concentration of the surface equipment	
Usually less mineral lost in pillars	Large and non-uniform pressure
Cheaper power supply	Less safety—two outlets to the surface instead of three
Both shafts can be used for hoisting material	Need for maintenance of the return unless it is driven in the rock
Shaft deepening is easier	
Air flow reversal is easier	
Better supervision of the operation of the fan	

The boundary layout requires a through path for the air from the central shaft to the boundary shaft. When the first horizon is being worked towards the boundary, the airways are driven as the faces advance; after the second horizon has begun to be mined, the former haulage road becomes the return airway. If the method of mining is then from the boundary towards the shaft, the haulage road must first be driven to the boundary of the take, which for a large area of coal may involve a considerable time. Fast methods of driving development roads, however, are eliminating this disadvantage of the boundary layout.

The "central-deviated" layout is obtained when a second, deeper hoisting shaft is sunk

Forced ventilation (Fig. III-1*d*) is convenient when there is some reason to expect the formation of cracks in the overburden, and settlement at the surface, for example, in the mining by caving of shallow thick seams or ore bodies; with exhaust ventilation a considerable amount of air would be sucked through from the surface, but as the mine deepens this leakage naturally becomes of diminishing importance, enabling forcing ventilation to be replaced by exhaust ventilation. This scheme is sometimes used for ventilating horizontal deposits (the Moscow coalfield) by central or boundary fans;

with forcing fans on the boundary every fan installation should be furnished with a heater plant.

The layout shown in Fig III-1*e* is used when the pressures of different sections are rather varied, and it is desired to avoid the installation of stoppings with regulators (see page 471), sometimes in this case underground fans may be used

The combined central-boundary layout (Fig. III-1*f*) is used where several producing sections are found in one wing; part of these are ventilated from a central fan and the others from a boundary fan

The layout of Fig III-1*h* is used in shallow mines, when the cost of sinking pits is low, in this case each panel is independently ventilated (the Estonian shale mines)

The boundary layout of ventilation with a fan installed at the surface for almost every section is widely used in the Kuznets Basin where the number of fans at each mine reaches 15 or more There is no general return air horizon at these mines The compelling reason for this ventilation layout is the extremely high coal yield from the coal measures, the seams being extremely thick and close together, in these conditions the maintenance of a general return air horizon is extremely laborious, in particular with any caving system

This method of ventilation has the following main advantages

(1) there is no need for a large number of ventilation structures for distributing the air flow,

(2) the head developed by each individual fan is small, which reduces both the power consumption and the air leakages;

(3) the airways can be of small cross section;

(4) safety is improved because of the large number of outlets to the surface,

(5) it is easy to isolate a section in an emergency

The disadvantages of the layout include

(1) the ventilation calculations are complicated,

(2) during production it is difficult to ensure the necessary air flow to each section and to completely exclude any interaction between the fans;

(3) the difficulty of reversal in case of emergency,

(4) the difficulty of supervision of the fan installations,

(5) the large number of men required to serve the fan installations (in case of changeover to automatic control, Items 4 and 5 above would be eliminated);

(6) the high cost of deepening and maintaining the ventilating pits to lower horizons,

(7) after an accidental fan stoppage, a change in the operating conditions of the fan is possible

At present in the mines of the Kuznets Basin considerable effort is being applied to change this layout for others, for example, a boundary layout with grouped fans, serving a group of close seams, or a layout with two central shafts

III-3 4 Surface or Underground Installation of the Main Fan

The Soviet Safety Regulations require the main fan to be installed on the surface because inspection on the surface is easier and more reliable, maintenance is easier and the power supply is cheaper; in case of fire or explosion, the fan on the surface can remain undamaged. Nevertheless there may be instances when an underground fan would be fully justified, for example:

(1) when the land around the ventilation shaft is slipping;

(2) when the land at the collar of the ventilation shaft is not convenient for building the fan structure or is subject to snow slides;

(3) in planning to change the working fan for a more powerful one there is found to be not enough space for it.

An underground installation has the following advantages:

(1) reduction of short-circuits through the collar of the ventilation shaft;

(2) possible saving in the cost of construction of the fan building, fan drift and diffuser,

(3) possible use of forcing fans without the installation of an airtight building;

(4) there is sometimes a substantial power saving when fans are installed separately for each wing or each horizon of the take (with the simultaneous mining of several horizons) when there is a large difference in the pressures in the air circuits through them;

(5) there is higher flexibility in control of ventilation when the air quantities flowing through the various sections of the mine must be changed.

In some countries, underground installation of the main fan is finding ever increasing application.

Thus, of the newly installed main mine fans in Holland the majority have been underground fans; similar installations have been put into service in Belgium, France and Germany. In these countries it is considered that an underground fan does not present a greater danger than a surface fan, provided that appropriate precautions are taken

A stand-by fan can be installed on the surface.

If the fan is underground, it is essential that

(a) the fan room shall be in the intake airway;

(b) the fan shall be remote-controlled for starting, stopping, and reversing.

The fan can be installed in a widened part of some airway with an air lock to allow men to pass through, or in a special fan room.

III-3.5 Exhaust or Forcing Ventilation

The Safety Regulations require all mines above methane Category 1 to be provided with exhaust ventilation. This requirement arose from the idea that after accidental stoppage of an exhaust fan, the air pressure increases by the magnitude of the fan head, and consequently for a short interval of time the gas emission from the worked-out area and open cracks in the coal will be delayed. In copper pyrite mines, forcing ventilation is preferred with top slicing methods of mining so as to drive into the worked-out area and the caved area the air which has been reduced in oxygen content, as well as to prevent the release into the stopes of heated air and combustion products.

Forcing ventilation is generally used in the Moscow coalfield to delay the release of CO_2 from old workings on the supposition (which needs confirmation) that under forcing ventilation less "gassing up" (large release of CO_2 from old workings) occurs. The use of an exhaust fan at one mine in the Moscow coalfield has not worsened the mine ventilation.

When a forcing fan is installed at the point where an exhaust fan was previously used, the direction of the air flow in the faces becomes descending which has unpleasant consequences. This disadvantage can be overcome by forcing the air into the lowest horizon; the mouth of the hoisting shaft then remains open and sometimes (if the intakes are dry) the air may not need heating.

III-3.6 Ventilation Planning in the Mining of Minerals Subject to Self-heating

Experience in the working of coal seams and pyrite ores subject to spontaneous combustion (self-heating) has shown that correct ventilation can greatly reduce the number of spontaneous heatings and underground fires.

The ventilation layout for working a mineral subject to self-heating must be such as to eliminate so far as possible:

(a) the leakage of air through pillars or through the waste from one working place into another;

(b) leakage of air from the surface*.

* Experience has shown that suction of air in from the surface more often starts fires from self-heating than leakage of air out of an underground working along cracks.

Leakages through the waste can be reduced, with advancing methods of mining, by using boundary ventilation, while with retreating mining methods, central ventilation should be used Leakages are greatly reduced by driving the developments in the stone In a shallow mine, to eliminate leakages from the surface it is sometimes advisable to use forcing ventilation.

Since any leakage of air takes place under the effect of a pressure difference, mines working a mineral which is liable to self-heat must have small pressure differences in places where air can leak through pillars or the waste, and the cross section of the airways must therefore be large.

Since a change in the direction of the air flow can cause self-heating, the ventilation should necessarily be artificial, and the fans must provide not only a constant direction of air flow, but also a constant air supply to each section This is difficult to achieve when many fans work at the same mine in opposition to each other as in the Kuznets Basin

In planning ventilation for pyrite mines we should remember that normal thermal conditions of work can be achieved mainly by ventilation. the air quantity must be enough; to prevent heating the intake air must have a short course; the ventilation of the faces and development roads must be by an active air stream.

III-3 7 Ventilation Planning at Very Gassy Mines

The main action against gas emissions in mines has hitherto been the active ventilation of underground workings. This measure is sufficiently effective and comparatively easy when there is a small gas emission for mines up to Category 3 inclusive and even for mines outside the categories if their gas emission is not very large, for example, up to 20-25 m^3/ton. However, there are mines where the gas emission climbs up to 80-100 m^3/ton.

With the standards accepted in the USSR for gas content (see Part One) and with the safety factor of 1.45 applied to air (see page 528) the air flow needed to dilute this gas amounts to 0.1 q m^3/min per ton for a section, and 0.135 q m^3/min per ton for the mine as a whole where q is the gas emission in m^3/ton For example, with $q = 75$ m^3/ton, a section output of 400 tons and a mine output of 3,000 tons, the air requirements are· 3,000 m^3/min for the section, and for the mine as a whole about 30,000 m^3/min.

The most effective way of reducing the gas emission is methane drainage, by which the gas emission from a section can be reduced by 50-60% or more and the gas emission of the mine as a whole by 35-45% (see Part One, Chapter 1); other measures directed towards the smoothing of the peaks of gas emission were considered in the

same chapter; methods of controlling gas emission in the face and in the worked-out area were also indicated, with various ventilation layouts used in mine section.

A mere increase of the air supply so as to dilute and remove the methane emitted in large quantities has the following consequences.

(a) the cross-sectional area of the main haulage roads becomes so large (up to 40-50 m^2) that the planned roadway becomes a pair of roads;

(b) the large air flow to the faces causes their length to be reduced and involves subdividing a face the length of the whole horizon into several faces (or sub-levels);

(c) because of the high pressure the air leakages increase considerably;

(d) the air moving at a high velocity raises the dust;

(e) the cost of ventilation increases.

Additional measures for gas control include.

(a) the lowering of the general gas emission underground,

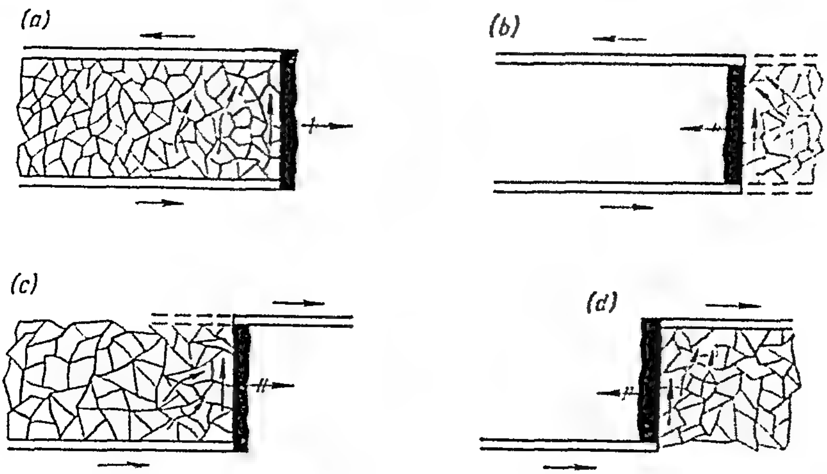

Fig III-2 Ventilation layouts for retreating and advancing

(b) the redistribution of the gas emission in time with a view to reducing it in places where work is being done, transferring it into locations where gas emissions are less dangerous, for example, the return airway (see Part One).

The redistribution of gas emission in space, and in particular its transfer from the coal face to the return airway, can be achieved by changing the direction of mining and the ventilation layout. Thus, for example, with high gas emissions from the wall rocks in the waste it is possible to greatly reduce the release of gas into the face if, with a central ventilation layout and advancing faces,

we allow enough air to leak through the waste to dilute the gas in it and push it directly out into the return airway (Fig. III-2*a*). A changeover to a retreating method of working leads to the following (Fig. III-2*b*) some gas from the waste will join the gas from the face with the result that the quantity of this gas increases; the air leakages will practically disappear. With the boundary ventilation layout and an advancing method of working (Fig III-2*c*) some air at the beginning of the face will pass through the waste, ventilating it,

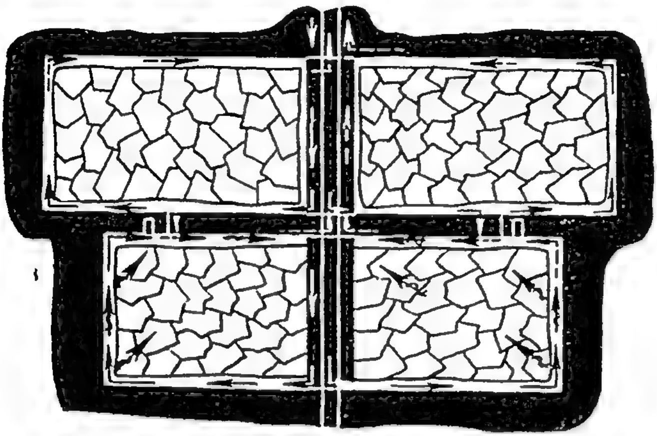

Fig. III-3 The method of adding intake air through the gate roads to the faces

and joining the air in the face at the top end; with a retreating mining method (Fig. III-2*d*) at the top of the face because of the leakages through the waste the air quantity will be less than at the bottom of the face, but, on the other hand, the gas from the waste will not reach the face. Thus, from the point of view of the improvement of face ventilation, the best layout will be (*b*) and the worst will be layout (*a*); layouts (*c*) and (*d*) (see Fig. III-2) will have an intermediate position.

The return air from individual faces is improved by introducing some intake air when the return air contains a considerable amount of gas; part of the intake air from the upper sub-level (Fig. III-3) is released through the stenton to the return airway of the lower sub-level.

III-3.8. Ventilation Planning for Mines Working Seams Subject to Sudden Outbursts of Coal and Gas

Ventilation planning should be preceded by a detailed survey of the deposit, which should indicate not only the location and amplitude of geological disturbances but also give information for

determining the coal strength, its gas content, gas pressure, the structure of the coal seams, etc With this information all the workable and unworkable seams should be divided into safe, menacing, and dangerous seams from the point of view of sudden outbursts Then considering that the safe seams will be extracted first as protective seams, the sequence of extraction of the seams should be worked out If there are no protective seams, mining should begin with the strongest, thinnest and simplest in structure It is necessary to decide also how many horizons will be simultaneously mined. From the point of view of safety, it is preferable to work two or even three horizons simultaneously; the topmost horizon will be used for production, the next horizon down will be used for mining protective seams and the third horizon will be used for the preliminary methane drainage of the dangerous seams.

At present there are the following ways of improving the methods of mining steep seams, which have the highest number of outbursts.

(a) mining by strips worked down the dip, working under a shield, and mining diagonally to the dip;

(b) manless mining of strips to the rise, with a face mined by a rope saw and with shotfiring from a stenton;

(c) mining along the strike with the horizon divided into sublevels and the upper sub-level ahead at a distance which is safe from the outburst point of view in the lower sub-levels;

(d) the use of cutter loaders with remote control.

In gently dipping seams, mining methods are developing in two directions, using cutter loaders with manless mining, and by drilling advance holes of large diameter toward the face with a view to removing methane from the seam being worked

Existing ventilation layouts both of coal mines and metal mines have been gradually developed through many years of mining practice These layouts have been adapted to various methods of mining, and to various methods of opening up and developing the deposit.

III-4. VENTILATION LAYOUTS FOR SEAM DEPOSITS

A THIN AND MEDIUM-THICK SEAMS

III-4 1 Gently Dipping Seams

Mining by Horizons. Depending on the method of mining, whether by an advancing or a retreating method, the following ventilation layouts are possible

1. With advancing longwall, the intake air comes along the haulage road, sweeping the face, and passing out by the return

airway (formerly a haulage road) reaches the upcast shaft The haulage road is supported either by coal pillars or by strip packs on each side; after the horizon has been worked out, the return airway is abandoned

2 With retreating longwall, because of the difficulties of maintaining the haulage road after the face has been mined out and the equivalent difficulties of repairing it after the transition to mining the horizon below, it is better for each new horizon to drive its own return airway, generally paired with another Recently, the achievement of high speeds for single haulage roads has made it possible to visualize long single drivages not driven in pairs.

Mining by Panels. The methods of ventilation are the same as for horizon mining, with the difference that the air flow direction indicated above is maintained within the panel only. The advantages of panel mining from the point of view of ventilation are as follows

1 The haulage and return airways are shorter and therefore the head lost in them is smaller.

2. The roads form a large number of splits, thus reducing the total aerodynamic resistance.

Other conditions being equal, the result is that, from the ventilation viewpoint, the coal output can be appreciably increased

Below are given some examples of ventilation layouts in thin, gently dipping seams.

The simplest layout has the face equal in length to the horizon (Fig. III-4), its disadvantage is the high air velocity in the face, which in very gassy seams limits the height of the horizon If the horizon is subdivided into sub-horizons (sub-levels), three ventilation layouts become possible

(a) series ventilation of the sub-levels (Fig. III-5),
(b) series ventilation with additions of intake air (Fig. III-6);
(c) ventilation of the faces by separate splits (Fig III-7).

In mines where descensional ventilation is authorized (see p 472) so as to avoid series ventilation of the sub-levels by the same air current, one of the layouts indicated in Fig. III-8*a*, *b* and *c* is possible.

Figure III-9 shows how, with series ventilation, the air flow passes from one sub-level to the next, and Fig. III-10 shows the method with split ventilation: the air from the top of the lower face passes along an airway through the waste downwards into the sub-level return airway, *a* indicates a stopping separating the air currents sweeping the upper and lower faces The other possible method of ventilating the upper rib of the face, by splitting from the intake of the upper sub-level, is less convenient because the rib will then be ventilated by a descensional current.

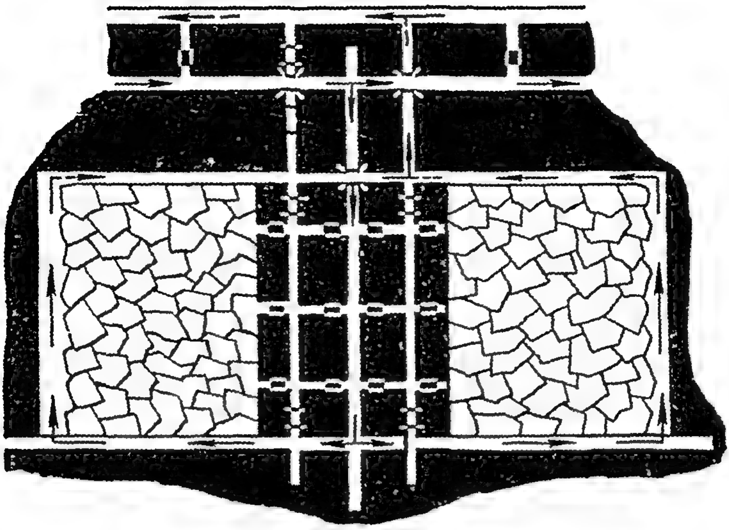

Fig III-4 Ventilation of a face the length of the horizon, in a thin, gently dipping seam

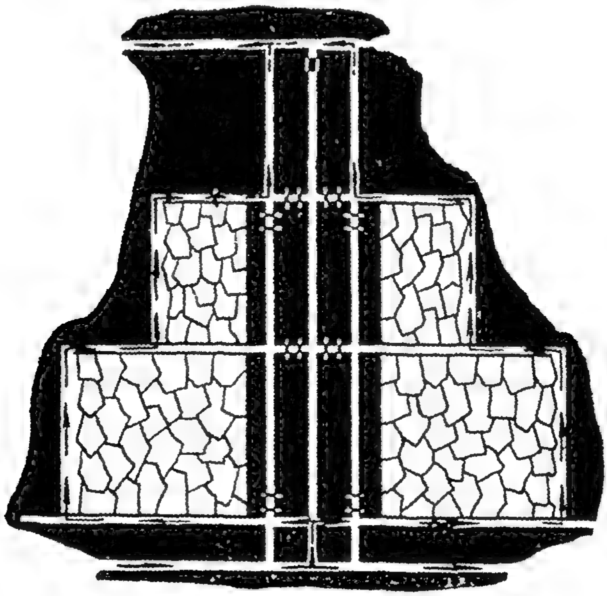

Fig III-5 Series ventilation of the faces in a horizon subdivided into several faces

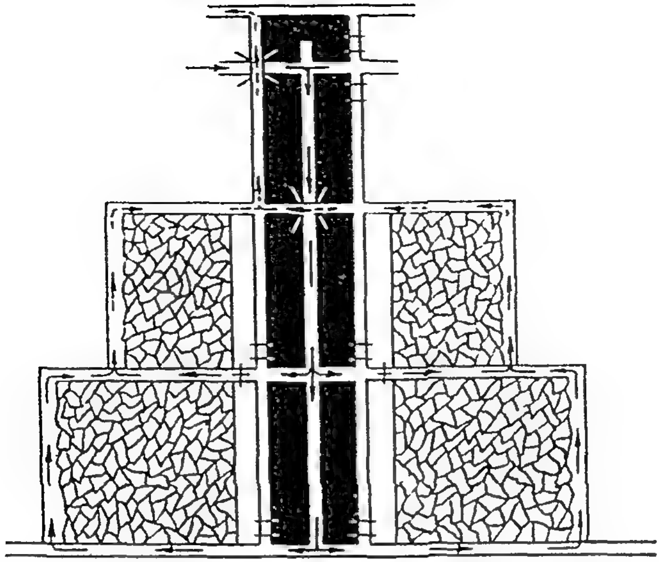

Fig. III-6 Series ventilation of the faces by adding intake air

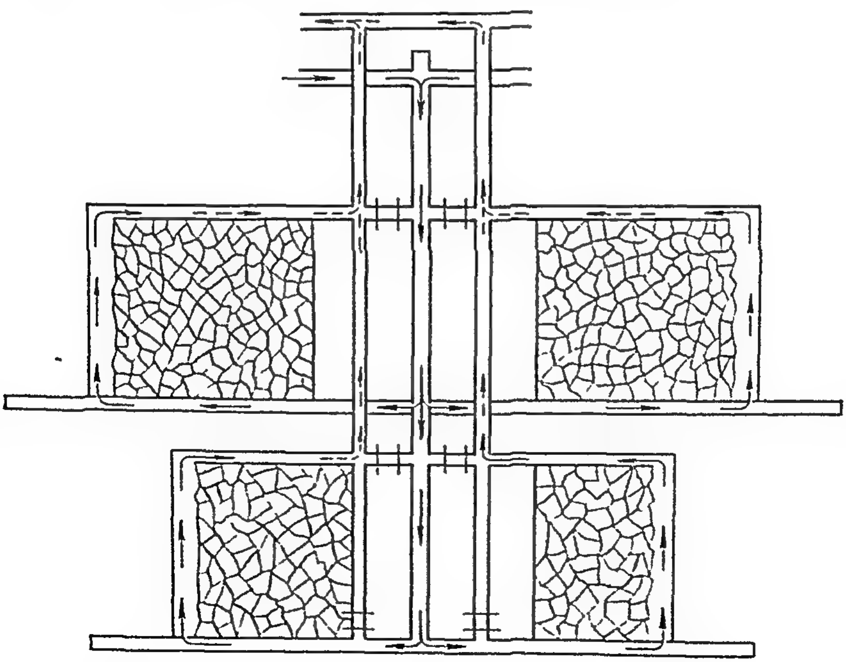

Fig III-7 Ventilation of the faces by splitting

Ventilation layouts with retreating faces are analogous to those used with advancing faces Part of the gas can be emitted during the development work in the drivages; the air leakages are small

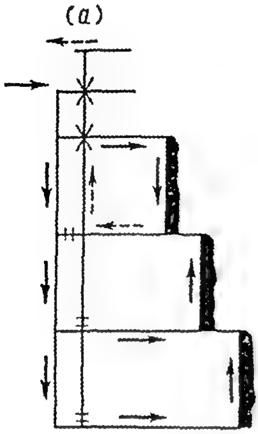

Fig III-8 Ventilation layouts for faces with descensional ventilation

since the roads are driven through coal pillars The disadvantages of retreating methods from the ventilation point of view are:

(a) the difficulty of ventilating the developments in gassy mines,

(b) the great length of the air circuit, increasing the resistance of a section

Figure III-11 shows a series method of ventilating the sub-levels,

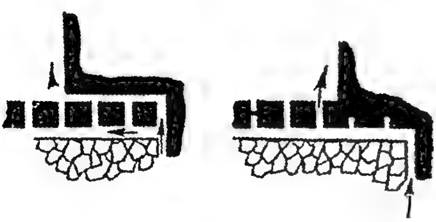

Fig III-9 Passing the air from one face to the next when the faces are ventilated in series

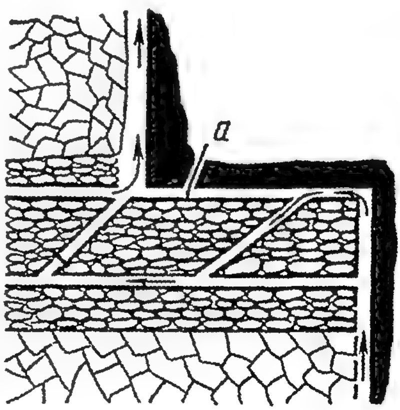

Fig III-10 Split ventilation of the faces

and Figure III-12 shows how they are ventilated by splits The latter method involves building eight air crossings when faces are being worked on both sides of the airways, and is therefore complicated.

Figure III-13 shows the transition from one sub-level to another. The two neighbouring air currents are separated by stoppings (*a*) in Fig. III-14 when each sub-level is ventilated separately. It is extremely difficult to achieve, with this method, complete air-tightness between one air current and the next one.

The developments (inclines and gate roads) are first ventilated by the layout shown in Fig. III-15. Later the layout becomes as shown

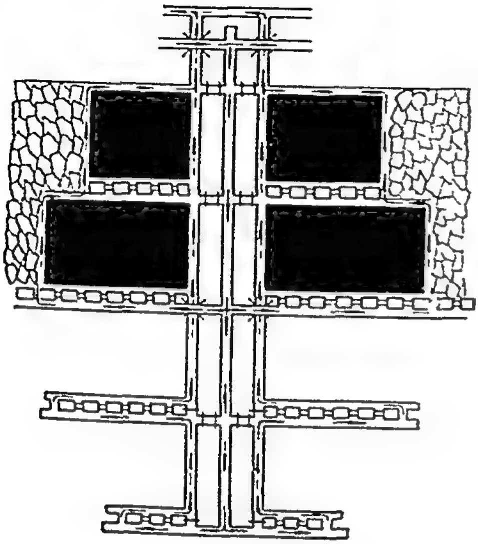

Fig III-11 Layout for series ventilation of the faces

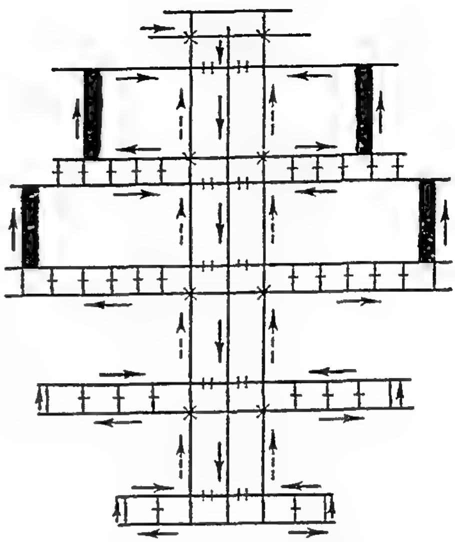

Fig III-12 Layout for split ventilation of the faces

in Fig III-16, with the faces of the developments ventilated separately, and finally, after the manway (parallel to the incline) has joined the return airway, as shown in Fig III-17. This diagram does not show the air crossings separating the intakes from the returns If the main ventilating stream is inadequate to supply the necessary amount of air, booster fans must be used.

III-4 2 Steeply Dipping Seams

The ventilation layouts used in the working of a single steeply dipping seam do not differ from those used in a gently dipping seam. Two methods of ventilating the face of the haulage road are shown in Figs III-18 and III-19. Figure III-19 is suitable for seams subject to sudden outbursts; when there is a fall of coal, the air passes to the face through the coal in the haulage road by a ventilation pipe.

When a series of seams are worked together by a section (or group) crosscut the following layouts are used

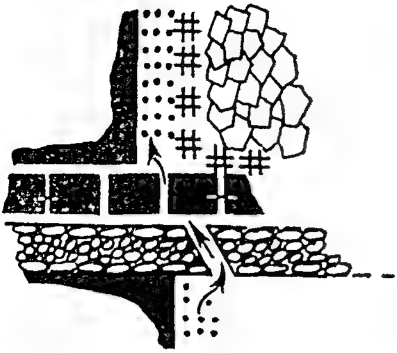

Fig III-13 Layout for transferring the air from one face to the next

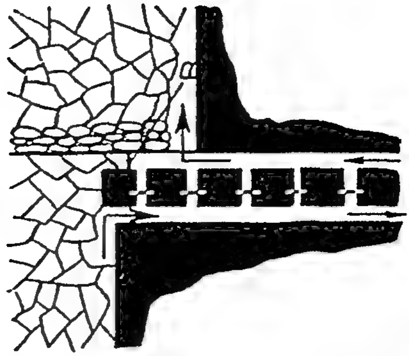

Fig III-14 Separation of two close air streams by stoppings (*a*) using split ventilation of the faces

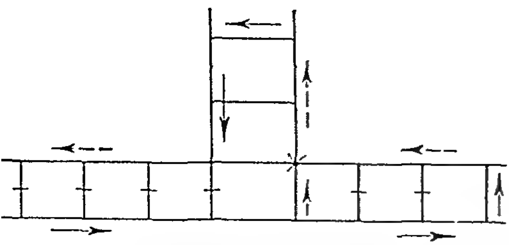

Fig III-15 Ventilation of the developments in the early stages

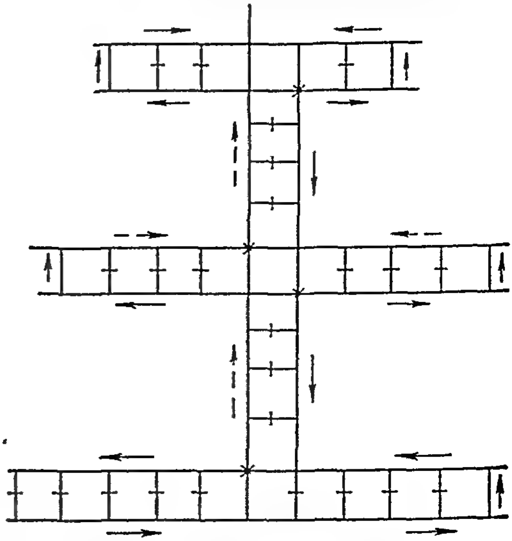

Fig III-16 Ventilation of the developments in the middle stages

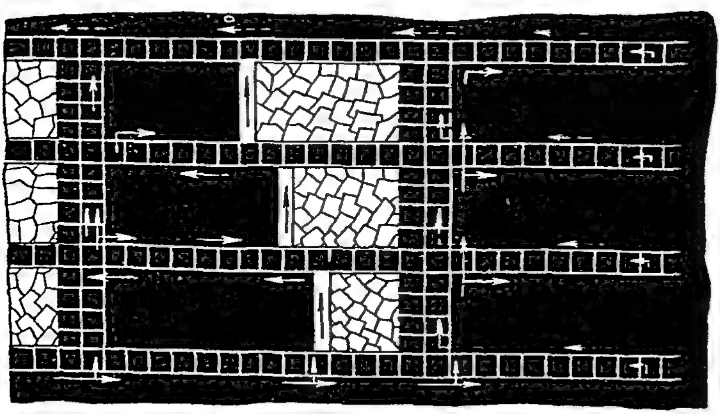

Fig. III-17 Ventilation of the developments in the production stage

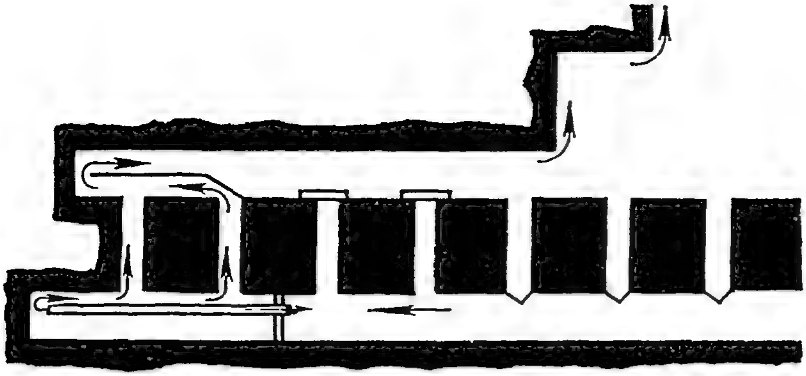

Fig III-18 Layout of the face ventilation of a haulage road in a steep seam

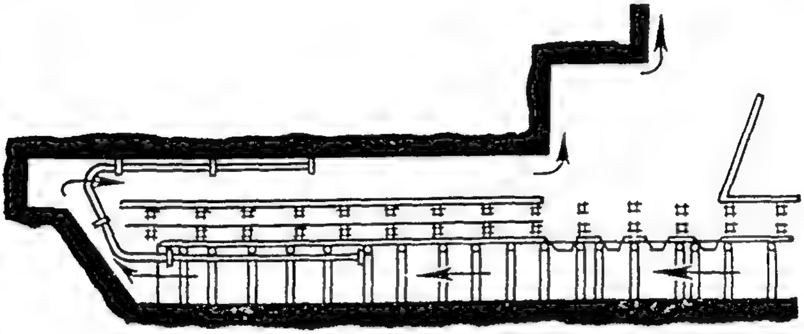

Fig. III-19 Ventilation layout for the face of a haulage road in a steep seam subject to sudden outbursts

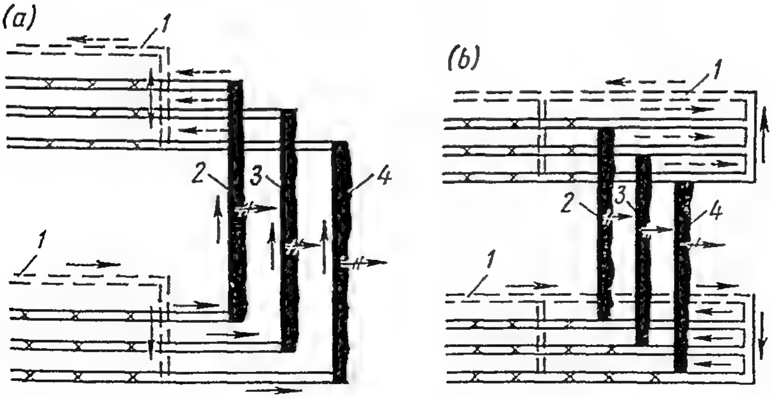

Fig III-20 Ventilation layout for mining a series of steep seams by advancing

(a) with haulage to the rear crosscut, (b) with haulage to the forward crosscut, 1—haulage road in the stone, 2, 3, 4—seams Abandoned airways are indicated by crosses

When the mining method is an advancing one, and haulage is through the rear crosscut (Fig. III-20*a*) the air from the group haulage road driven in the stone passes through the group crosscut into the gate road in the seam; after passing along it and sweeping the face, the air leaves the return airway and passes into the main return in the stone. Coal pillars are left to support the return airway in the seam.

The ventilation layout used when haulage is through the forward crosscut is indicated in Fig III-20*b*; this differs from the previous scheme by the reverse flow of the air along the gate roads in the seam.

To ensure an effective air flow at the start of the production face of the lower horizon (before the mining out of the upper horizon) a return airway must be driven to pass the return air from the lower horizon

When the method of mining is a retreating method, the return air passes into the repaired, former haulage road of the upper horizon, and then along the intermediate crosscut into the group return airway driven in the stone. If, however, this road must still serve as an intake for the upper horizon, a return drift will have to be driven in the stone to remove the return air from the new horizon, by-passing the upper horizon.

III-4 3 Ventilation Layouts for Mining by Brake Inclines to the Rise and Haulage Inclines to the Dip

These layouts are shown in the earlier illustrations and do not require further explanation (Fig III-5 is a brake incline take, Figs III-11 and III-19 are dip takes) The main feature of these layouts is that the intake air and the return air streams (along the brake incline or the haulage dip and their parallel manways) are driven close to each other for long distances with the result that air leakages occur through the numerous stoppings, doors, etc, reaching considerable quantities, and the faces are not adequately ventilated To reduce these losses, pillars of adequate thickness must be left (e g 20 m) and plastered masonry stoppings must be built in the stentons, with double doors to separate the air streams To reduce air losses the inclines should be driven in the stone

B METHODS OF VENTILATION IN THE MINING OF THICK SEAMS

1. Methods of Mining by Inclined Slices. The ventilation layouts are adapted to the mining method to be used. The air passing along the haulage road (usually driven in the stone) reaches the section crosscut (whether horizontal or inclined), then along the inclines in the coal rises into the slice being mined, sweeps the face, then

passes along the slice return airway into a stenton and into the return air crosscut and out by the upcast shaft Within each slice the layouts are almost similar to those used for thin seams

Figure III-21 shows the ventilation layout for development by roads in the stone As can be seen, the air from the roadway in the

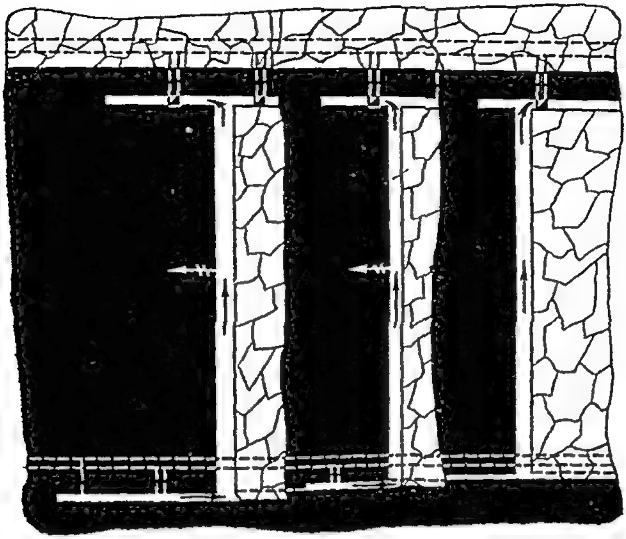

Fig III-21. Ventilation layout with development by stone drifts

stone passes along an incline up to each slice, sweeping it, and then along an inclined return airway up to the main return in the stone

2. **Mining by Shield.** The ventilation layout of the shield method of mining is shown in Fig. III-22 The air passes up the manway compartment of the nearest raise in the coal to the pillar being worked, up into the coal face under the shield, sweeping along the whole underside of the shield and flows through the opposite raise into the horizontal stenton and then into the manway compartment of the raise of the neighbouring pillar. Along this raise the air rises to the "minus" return airway driven at 4-5 m below the former haulage road of the mined-out horizon, passing out of it along an inclined crosscut driven every 200-400 m

The intake and return airways are connected by boreholes of 300 mm diameter driven upwards every 40-50 m. After the boreholes have been driven by augering machines, they are widened to the required cross section. They ventilate the face of the upper road by the fan installed in the haulage road; a second fan ventilates the face of the haulage road.

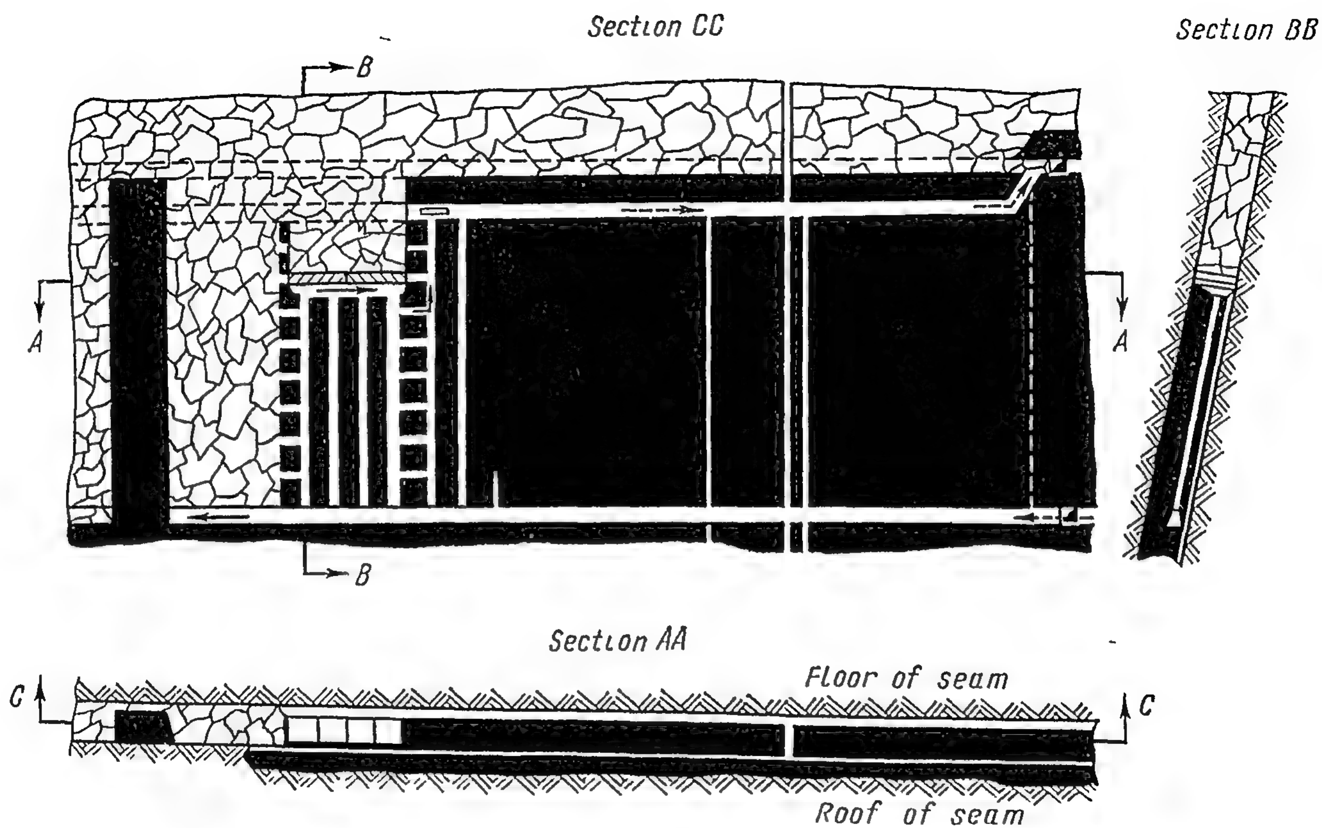

Fig III-22 Ventilation layout for the shield method of mining a steep seam

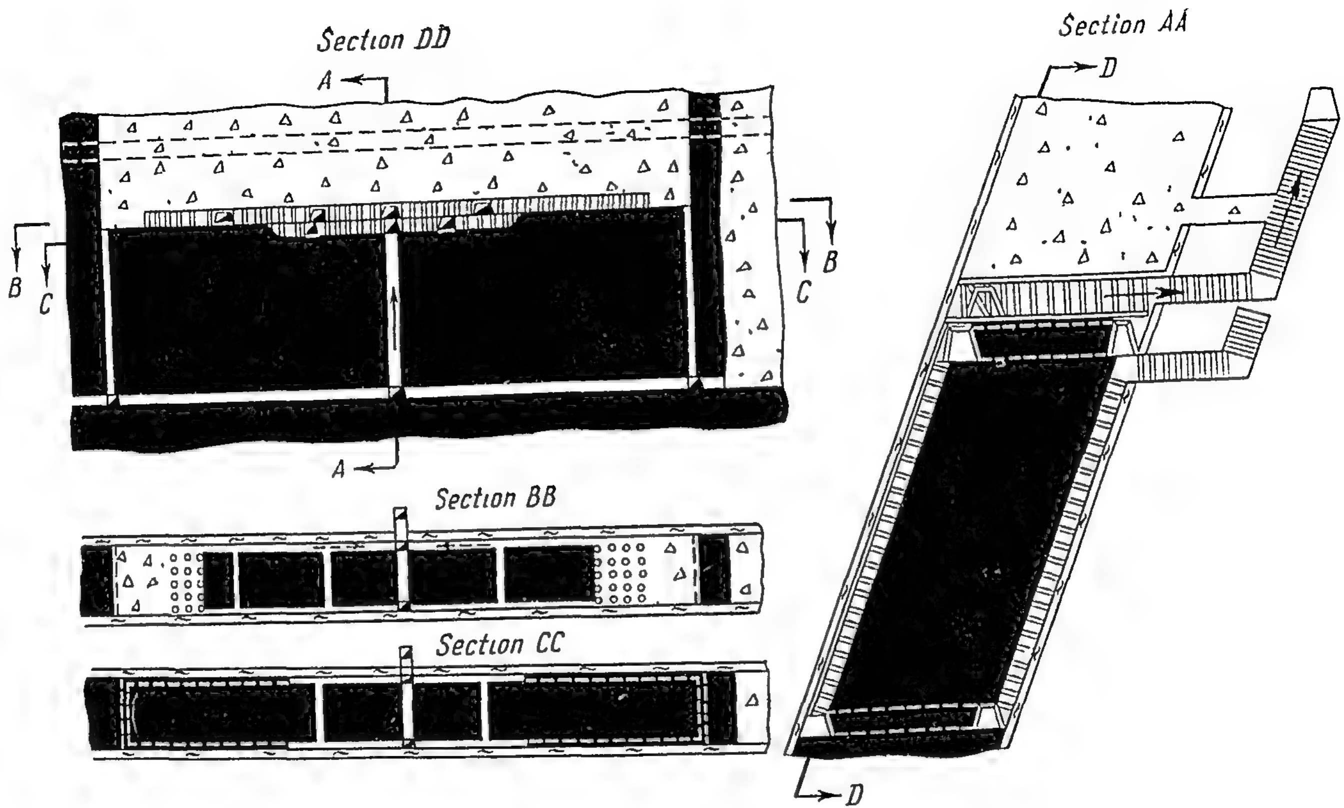

Fig III-23. Ventilation in mining a seam by horizontal slices

3. Mining by Horizontal Slices. Ventilation with this method, mining the slices parallel to the strike, is as follows (Fig. III-23). from the haulage road (whether in the stone or a group haulage road in the coal) the air passes along the group crosscut into the seam being mined, and then along one or two raises into the slice being worked, then into the slice road, and sweeping the face, returns to the middle of the section along the second slice road and passes along the central packing road into the return air crosscut The slice roads are connected every 50 m by raises. If the direction of mining is across the strike, boundary raises must first be driven along the edges of the section to provide an active air current into the slice

C VENTILATION LAYOUTS IN HORIZONTAL SEAMS

Figure III-24 shows the ventilation layout adapted for mining the shales of Gdov by double-unit faces, the air passes along the

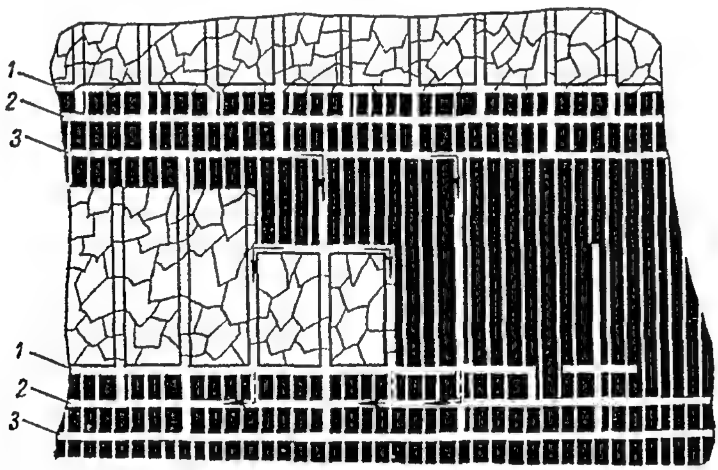

Fig III-24 Ventilation layout for mining a horizontal shale seam by double-unit faces

1—development roads, *2*—panel return airways, *3*—panel haulage road

panel haulage road into the conveyor road, where it splits into two and sweeps the faces; from the faces the air passes along the return airways in the coal to the panel main return airway Booster fans (see p 358) of "Prokhodka-500" or BM-200 type are used to improve the ventilation of mine sections which have too little air.

Figures III-25 and III-26 show two ventilation layouts used in the Moscow region coalfield

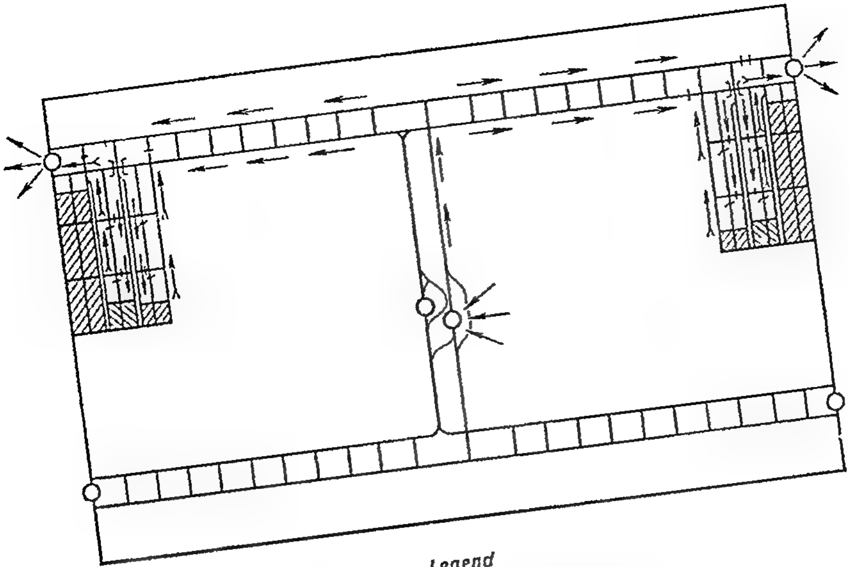

Fig. III-25. Ventilation layout used in the Moscow region coalfield

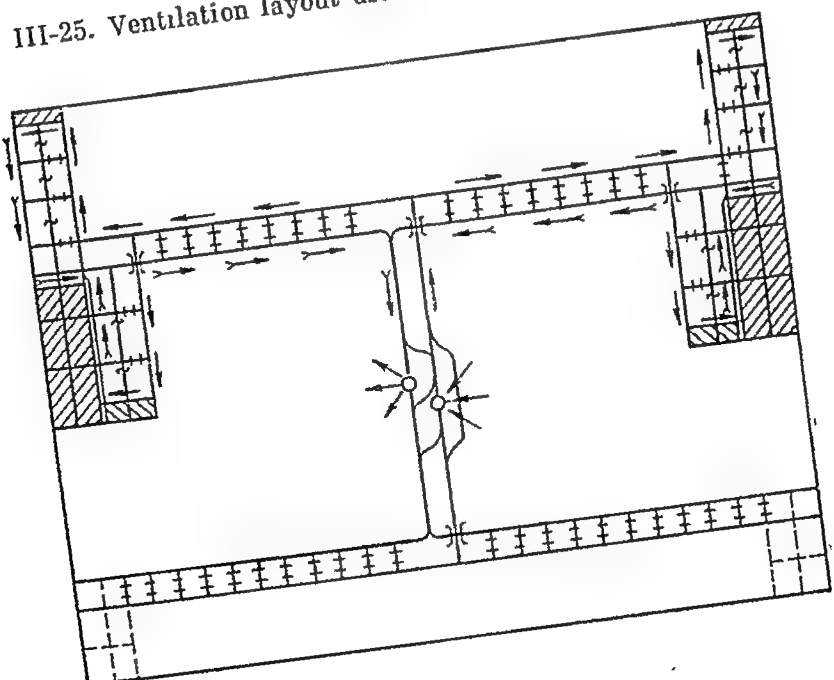

Fig. III-26 Ventilation layout used in the Moscow region coalfield

D VENTILATION LAYOUTS FOR SIMULTANEOUS MINING OF SEVERAL HORIZONS

When work on one horizon has finished and work is beginning on the next horizon below it, the need arises to provide intake air to two horizons Unfortunately this simultaneous working of two or sometimes more than two horizons is not exceptional either for coal mines or for metal mines Observance of the requirements for separately ventilating each horizon greatly complicates the ventilation layout because the need arises at each horizon, except the

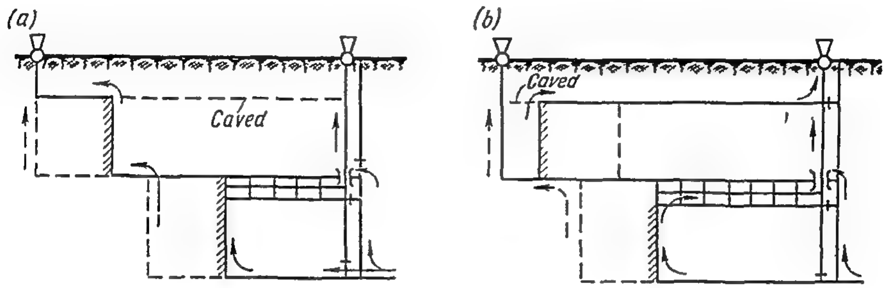

Fig III-27 Layout for split ventilation of two faces
(a) the upper face advancing, (b) the upper face retreating from the boundary

highest and lowest, to have two airways, a return and an intake, the intake for ventilating the upper horizon, and the return for taking the air from the horizon below.

This can be achieved by one of the two methods:

(1) the return air can pass out along special raises to a return air horizon which is common for all horizons,

(2) the return air passes along each horizon directly to the upcast shaft

In the first method, blind pits are used for taking the return air, as well as chutes, manways, raises, or old inclines, and the air passes through one or more horizons to the topmost main return horizon, at intersections with the intake airways, air crossings must be built, sometimes these raises are driven specially for ventilation

In the second method, the return air from the lower horizon passes along return airways driven together above or below the haulage road of the upper horizon

As an example of the ventilation of two horizon faces by splitting, two ventilation layouts are given, in Fig. III-27*a* showing an advancing mining method, and in Fig III-27*b* showing a retreating mining method A return airway driven in a coal pillar takes the return air from the lower horizon; after mining of the upper horizon

has reached the boundary (Fig. III-27*a*) or the two faces have "crossed" (Fig. III-27*b*) there is no longer any need for this return airway, and the return air from the lower horizon can pass along the haulage road into the boundary airway.

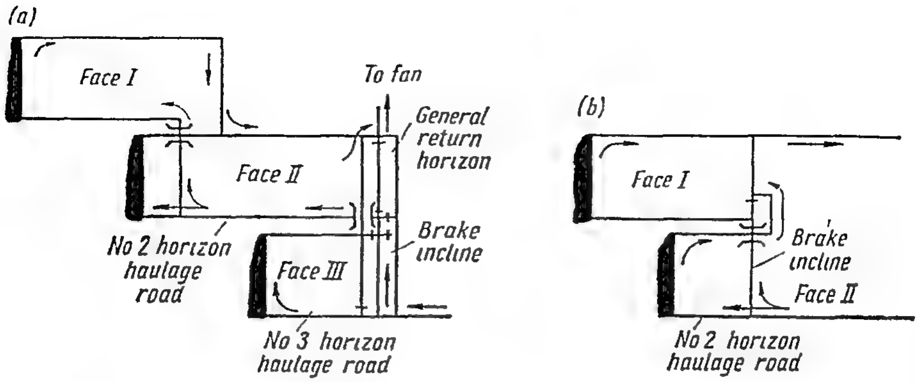

Fig. III-28 Examples of split ventilation layouts for several faces

Other examples of the use of splitting for face ventilation are given in Figs. III-28*a* and *b*.

Separation of the intake and return airways at the same horizon can also be achieved by dispersal of the coal faces, e.g. when at one

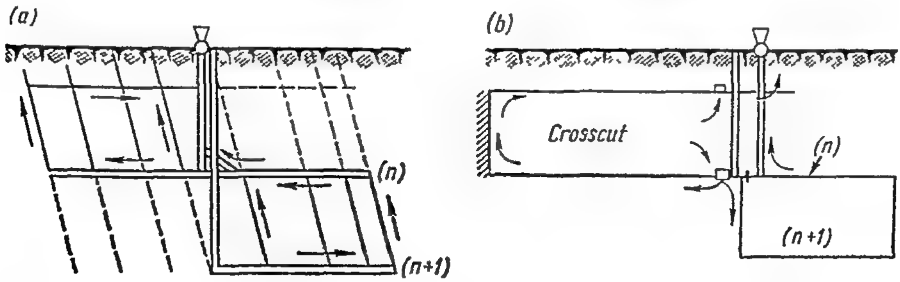

Fig. III-29. Layout for separating the intake and the return airways by dispersal of the faces

horizon (n) some seams are being mined, and at the next lower horizon ($n + 1$) other seams are being worked (Fig. III-29*a*) or when the same airway serves at one point as a haulage road and at another point as a return airway (Fig. III-29*b*).

When there are a large number of horizons (5 or 6) being worked simultaneously, it is sometimes convenient to divide the whole deposit along the dip into two completely separate ventilation sections.

III-5. VENTILATION LAYOUTS FOR ORE DEPOSITS

Ventilation layouts for ore deposits have special features arising from the differences in mining methods as compared with seam mining and from differences in the safety regulations

(1) with some methods of mining by caving, the face is not ventilated by an active air stream,

(2) downward air flow in the stopes is permitted, as well as the use of return air for ventilation along part of its course;

(3) auxiliary fans are widely used for ventilating stopes;

(4) recently, the removal of the return air has been effected by special return airways driven in the stone

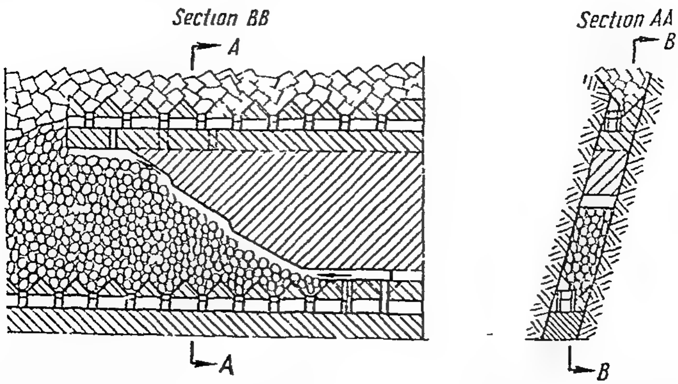

Fig III-30 Ventilation layout for mining a thin steep vein

In view of a large variety of methods of mining ore deposits, only those are considered below which have the most typical ventilation layouts, including bulk caving methods.

1. *Mining thin, steep veins by timbering, or packing, or shrinkage stopes.* Layouts for ventilating stopes with a return air horizon are extremely simple (Fig III-30). along one of the boundary raises, or along one of the short raises, the air rises into the stope, sweeps it and passes up to the return airway The ventilation is good.

2. *Open stopes with regular ore pillars in mining an inclined or horizontal deposit* (Fig. III-31). The air passes out of the haulage drift in the rock on one side of the stope, through one or more ore chutes into the stope sweeping it and passing out along other ore chutes into the parallel drift in the rock. The conditions of ventilation are not excellent because of the distribution of the air in the large open stope. For more effective ventilation, small portable fans should be used, not permanently installed at stoppings, close

to the places where the air is stagnant, and where the air must be removed to take away dust or fumes from blasting Entry to the stope before work starts in the face is by the ore chutes.

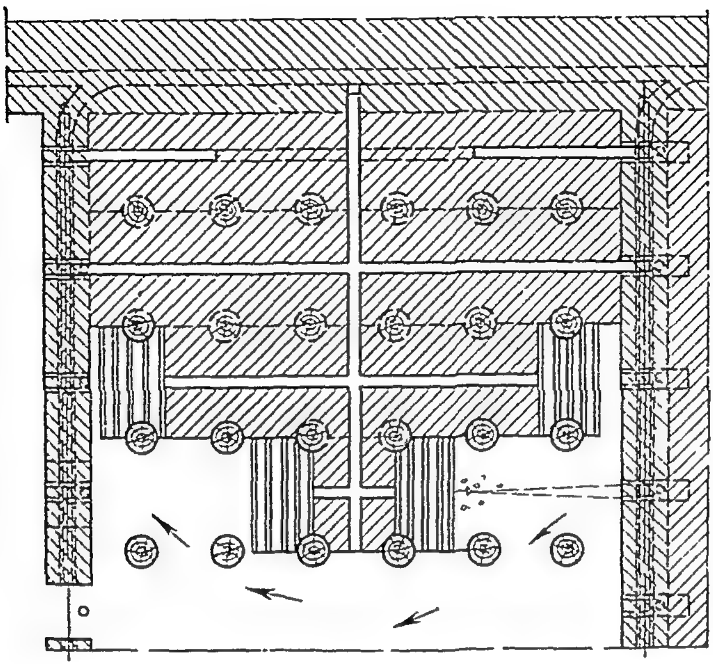

Fig. III-31. Ventilation layout for mining by open stopes leaving regular ore pillars

3. *Mining thick deposits by shrinkage stopes.* The ventilation layout is shown in Fig III-32. The air entering along the haulage road *1*, driven at the hanging wall, passes up the raise *2*, to the next manway *3*, along it to the stope *4*, sweeping it and then along the central raise *5* out into the main return airway *6*. The ventilation is good, being effected by a through air current.

4. *Top slicing.* The ventilation with this method is usually unsatisfactory.

When there is only one ore chute (raise), effective ventilation of the stopes by a through current from the main fan is impossible and auxiliary fans must be resorted to; even with two raises, dead ends which are ventilated only by diffusion are inevitable. The return air drops to the haulage horizon and passes to the next stope where it becomes even fouler. If the safety regulations requiring cleaning of the air are not fulfilled, the stope will contain a large amount of carbon dioxide, explosive fumes, and dust, and in the mining of a pyrite deposit will often be overheated.

Series ventilation of stopes can be avoided when there is no ventilation horizon if return air from each stope is directed

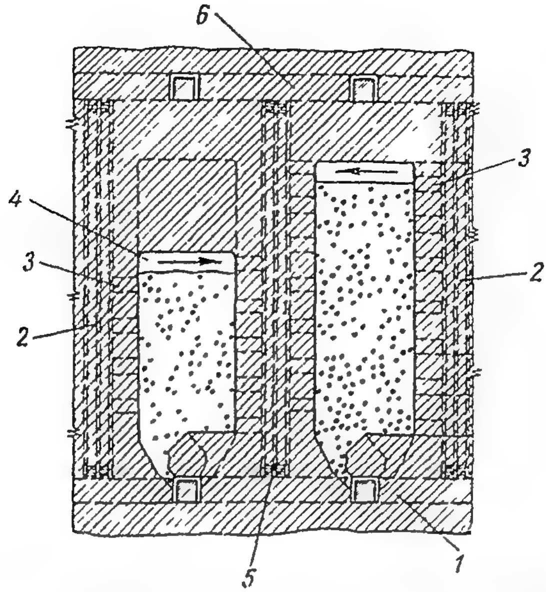

Fig. III-32. Ventilation layout for mining thick deposits by shrinkage stoping

either along ducting or through special ventilation raises to the return airway driven parallel to the haulage road or slightly above it.

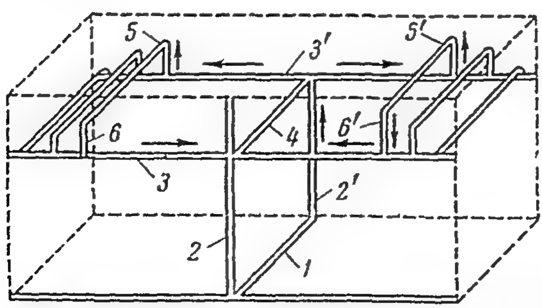

Fig III-33 Stope ventilation layout proposed by the NIGRI Institute at Krivoi Rog

The Geological Research Institute at Krivoi Rog has proposed the ventilation layout of Fig. III-33 to ventilate stopes effectively. In the middle of each stope a crosscut *1* is driven, and from it two raises *2*, and *2′*, one of them in the ore at the hanging wall, the other in the rock at the footwall. Along the footwall and hanging

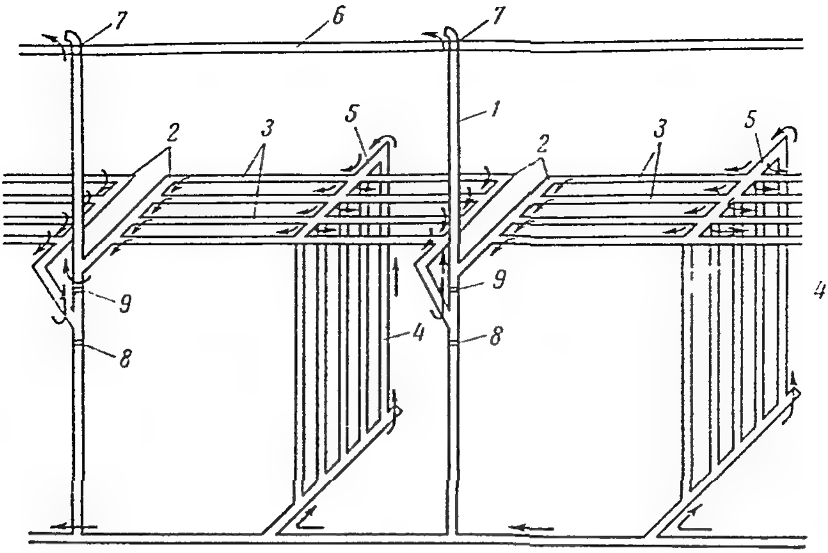

Fig III-34 Ventilation layout for sub-level caving

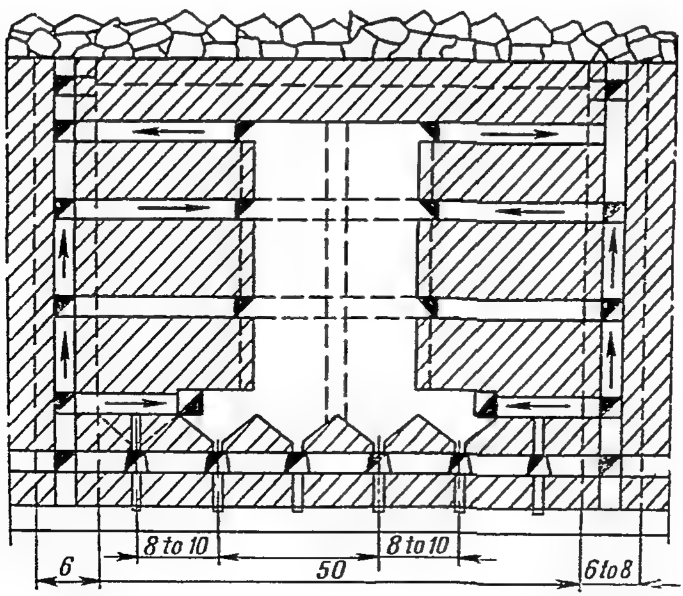

Fig III-35 Ventilation layout for sub-level stoping

wall contacts, two slice roads *3* and *3'* are driven, at every fourth slice, joined at the centre of the stope by the slice crosscut *4*.

The intake air from the haulage drift in the rock passes into the crosscut *1*, to the raise at the hanging wall, and up into the slice ventilation road where it splits into two streams, rising up the short ventilation raises *5* and *5'* at the boundaries of the stope, and along it into the drifts working the ore; then the streams flow along the ore chutes *6* and *6'* into the scraper drift, and then into the raise driven in the rock along which it passes out to the upper return airway not indicated on the drawing.

5. *Sub-level caving.* Ventilation with sub-level caving is somewhat easier than with top slicing, particularly when there is a return

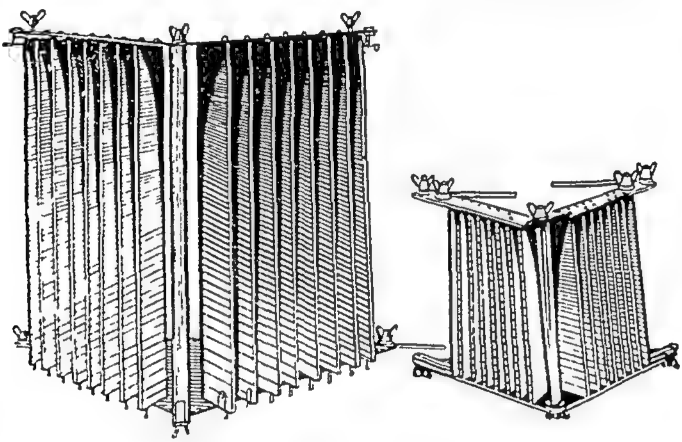

Fig III-36 Two-sided "turbulent fitting"

airway in the rock Figure III-34 shows the ventilation layout recommended by M. M Boguslavsky. Half-way between the stope centre lines, a raise *1* is driven in the footwall. From this raise at each horizon, two crosscuts *2* are driven at 4-6 m from each other; these crosscuts join all the sub-level drifts *3* The intake air entering the stope ventilation raise *4* (driven in the hanging wall) passes up to the sub-level crosscut *5*, from which it flows along sub-level drifts, sweeping the stope, and then to the sub-level ventilation crosscut *2*, to the inter-stope raise *1*, the return airway *6*, and then to the surface.

To improve the ventilation, auxiliary fans *7*, *8* and *9* are installed in the upper part of the inter-stope raises at trapdoors in the raise *1*. The air is distributed along the sub-level drifts by openings not shown in the drawing.

6 *Sub-level stoping.* The ventilation with this method is considerably better than in top slicing and sub-level caving (Fig. III-35).

However, when the stopes are large, the dilution and removal of the fumes in the stopes from blasting is very slow.

To speed up this process, F. A. Abramov (the Dnepropetrovsk Mining Institute) proposed the installation of special brattices to increase the turbulence of the air at its point of entry into the stope; the method was tested by G. V. Duganov and found efficient.

Figure III-36 shows a two-sided guide vane brattice; it consists of 16 vanes, 8 on each side, with an aerofoil cross section like the blades of an axial-flow fan. The value of the air flow structure factor with this fitting rose to 0.13 with the vanes at an angle of attack of 25°, shortening the time needed for ventilation by 30% (see Appendix 6)

The scraper horizon is ventilated by the air passing to it along the chutes

III-6. VENTILATION LAYOUTS FOR MINING METHODS USING HEAVY BLASTING

The main feature of these methods is the heavy explosive charges used in large open stopes; at the foot of the stope are funnels, connected by draw raises to the drift for scraping or breaking the rock; along these draw raises a small quantity of air can pass into the stope.

If the drifts from which the holes are drilled are blasted, an effective through current of air will pass along the sides of the stope, after the blast, if they remain intact there will be no through flow of air In the former case the ventilation layout must provide for intake air to the stopes. If the drilling drifts are not blasted, the intake air is supplied to the scraper or crushing drift to dilute and remove the explosive fumes passing from the stope into the drift, as well as the gas and dust formed when the oversized blocks are blasted.

Below are considered some of the more typical ventilation layouts used in mining methods involving heavy blasting.

1. *Sub-level stoping with long horizontal holes* (Fig III-37). The air passes out of the haulage drift in the rock *1* in two splits to the haulage crosscut *2* and the crosscut *10*. The first stream passes up the draw raises *3* into the crosscut *4* at the horizon from which the ore is drawn, ventilating it, and then flows up the raise *5*, along the drift *6*, into the raise *7* in the rock and up to the return drift *8*. The drilling drifts located at the hanging wall are ventilated by the air passing along the second raise in the ore, not shown on the drawing. If the drilling drifts have been blasted and if they are located on both sides, the second stream, passing along the raise *9* in the ore, enters the stope *11*, sweeping it and creating an effective through current in the stope. The return air passes along the raise *7*

in the rock and then out to the return horizon and the upcast shaft. If the drilling drifts are not blasted, the gases in the stopes, after the ore has been drawn, will pass out along the horizon at which the ore is drawn, only by seepage through the broken ore.

2. *Block caving* (Fig. III-38). The intake air from the main haulage road passes into the crosscut and from it along the raises *1* and *2* to the scraper horizon where it disperses along the crosscuts *3*,

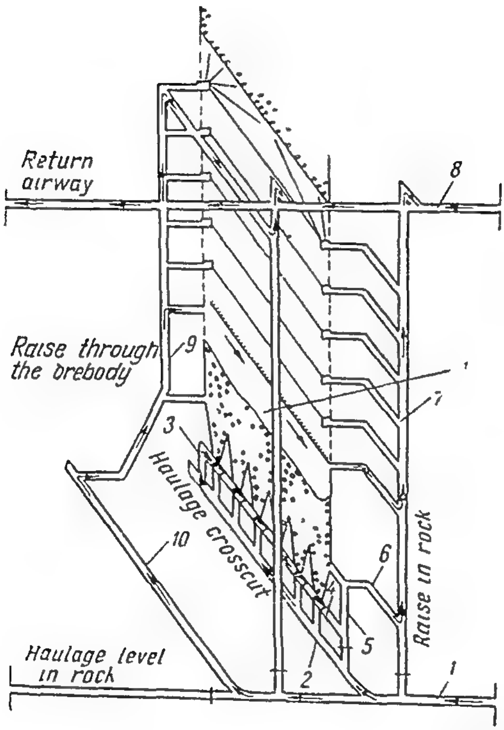

Fig III-37 Ventilation layout for the stoping of a massive deposit by long horizontal holes

sweeping them and passing, from both sides, to the central ventilation level *4*; the air then passes to the boundary crosscut *5*, and then up the raises *6* and *7* to the return air horizon. The remainder of the air sweeps the boundary drift *8*, and then passes along the raises *9* and *10* to the return air horizon. When the central ventilation level *4*, under the effect of heavy rock pressure, has to be abandoned early, it is driven 4-6 m below the scraper horizon and is connected to it by short raises

3 *Induced block caving* (Fig III-39). The intake air from the two airways, *1* in the rock and *2* in the ore, enters by two crosscuts *3*,

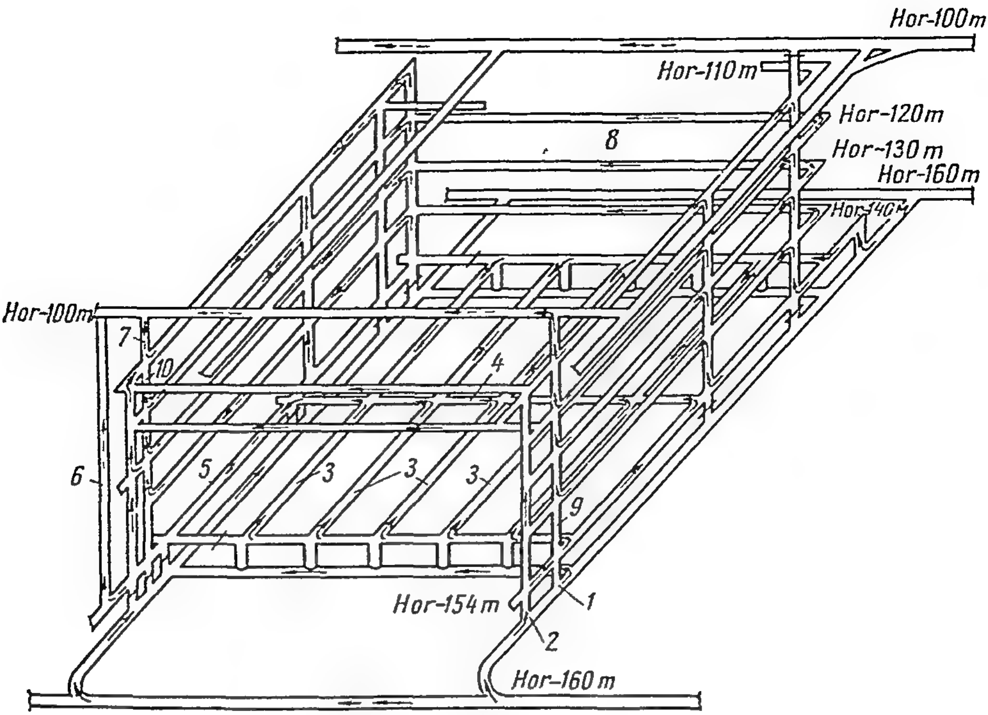

Fig III-38 Ventilation layout for block caving

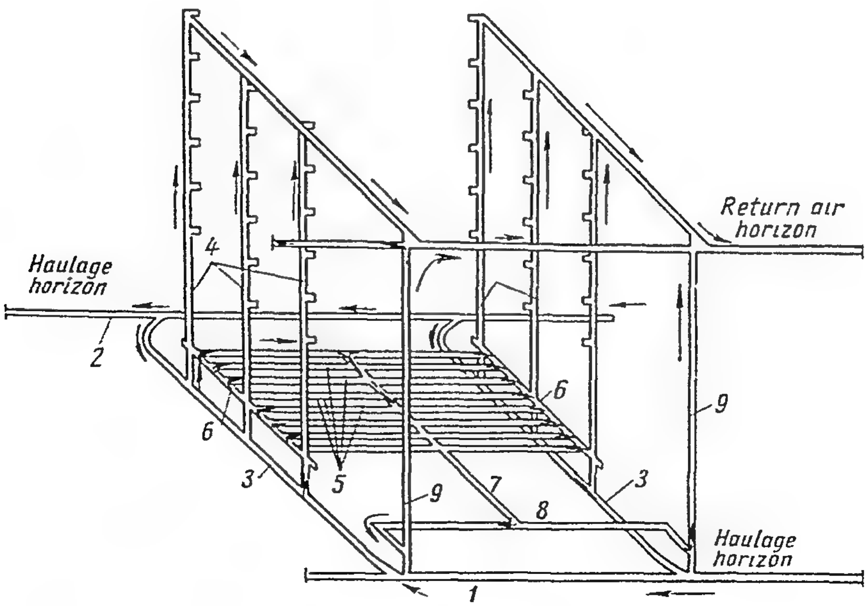

Fig. III-39 Ventilation layout for induced block caving

and passes from them along the raises 4 to the scraper horizon 6, where it disperses among the drifts 5. The remainder of the air sweeps the drilling drifts in contact with the raises. The return air is collected in the gathering crosscut 7, and passes along it to the level 8, at the footwall, and then flows along the raises 9 driven in the rock up to the return air horizon.

III-7. CALCULATION OF THE AIR QUANTITY REQUIRED FOR VENTILATING A MINE

For a correct calculation of the overall quantity of air needed for a mine, it is necessary to have a clear idea of the purpose of ventilation and of the function of the air passing into the mine.

The purpose of ventilation can be briefly expressed as follows: by providing the necessary quantity of air to create safe and healthy working conditions, ensuring a high output. The functions of the air sent underground include

(a) to ensure clean air for the men in the mine;

(b) to dilute and remove from the airways the various explosive and harmful gases or dusts which are emitted or formed underground;

(c) to cool the men working in deep mines so that they can give a high output

Depending on the type of mineral being worked and on other factors the quantity of air required underground is considerably affected by its function in gassy mines the quantity of gas emitted is important; in metal mines, usually the volume of blasting or the quantity of airborne dust is the deciding factor.

Although the purpose of the mine ventilation and the functions of the air are perfectly clear, its quantity is not easy to calculate, and final solutions do not yet exist.

The difficulties encountered in calculations of the air quantity include·

(1) the difficulty of early determination of the quantities of gas and dust which will pass into the mine air at various stages of the work,

(2) the even greater difficulty of determining the expected leakages of air,

(3) inadequate knowledge of climatic conditions in mines, gas emission, and the processes of dilution and removal from the mine of the various gases and dusts

Mine ventilation is unfavourably affected not only by insufficient air but also by excess of air; if the flow increases, the excess air stimulates leakage and self-heating of coal and pyrite ores, raises the dust and carries it about the mine; in addition, the excess air exerts a higher cooling effect, and the cost of ventilation increases.

All these factors indicate the great importance of determining the correct quantity of air flow for the mine as a whole.

These various functions of the air have found their expression in the form of certain standards in the Safety Regulations, controlling the quantity of impurities in the mine air (see Part One), including the air quantities per man and per ton of output. Nevertheless, even using these principles, the air quantity can only be accurately calculated for working mines; for mines being planned the whole calculation is purely tentative.

We must note here a further difference between coal mines and metal mines in their air requirements. In coal mines, despite the non-uniform gas emissions in the production faces during the various shifts, the total gas emission of the mine has been shown by experience to change little, and consequently the air requirement is constant. In metal mines, with their greater volume of blasting, the air requirement varies from a minimum in the intervals between blasting to a higher level usually at the end of the shift and a maximum after heavy blasting.

At present the following methods of calculating the air quantity are known, based on.

(1) the greatest number of men underground at any one time,
(2) the quantities of gases actually liberated;
(3) the mine output;
(4) the consumption of explosives;
(5) dust formation.

Sometimes (in deep mines) the quantity of air has to be calculated with a view to its cooling effect.

The quantity of air can be calculated: (a) for the mine as a whole, and (b) as the sum of the air quantities Q_1, Q_2, Q_3, etc., needed to ventilate the different seams, sections, etc The second method is more accurate but more laborious

1. *The calculation of the air quantity by the number of men underground.* The standard quantity of air required per man according to the Safety Regulations is established at 6 m^3/min, consequently, if the greatest number of men working underground at any one time is equal to n, the air quantity required is.

$$Q = 6n \text{ m}^3/\text{min} \qquad \text{(III-1)}$$

In fact both in coal mines and in metal mines, usually more than 6 m^3/min are provided per man. This method is the least suitable because men are not the main "consumers" of air. During heavy work the air needed by one man is only 60-75 litres provided that the air is continually renewed.

With the increase of mechanization and the reduction of the number of men occupied underground, the air standard per man

continually increases and at present, as pointed out, it is 6 m³/min. Underground observations have shown that if no other sources of impurity in the air are present, the air requirement per man is 750 litres, preferably 1 or 2 m³. The difference between the need and the standard is explained by the fact that in the total balance the oxygen requirement of man underground occupies a modest place.

2 *Calculation of the air quantity by the volume of gases emitted.* Let the amount of gas emitted in the mine in 24 hours be q, which must be diluted to the content $p\%$, then the required air quantity will be

$$Q = \frac{100\,q}{p \times 24 \times 60 \times 60}\ \text{m}^3/\text{sec} \tag{III-2}$$

Subject to the condition that the gas measurement is accurate this method is the most suitable.

At present there is in the draft stage in the USSR a document "Provisional Instructions for Calculating the Air Quantity Required in Coal Mines", designed for drawing up ventilation projects for working and planned mines. According to this instruction the air quantity is calculated.

(a) for working mines, according to the actual gas emission determined by the "mining statistical" method in which the methane emission at a greater depth than the existing workings is forecast for a deepening not greater than 100 m vertically for methane emission gradients up to 10 m/m³/ton, and up to 200 m for gradients exceeding 10 m/m³/ton;

(b) for mines being planned, according to the gas emission determined from the gas content of the coal seams which have been found in the geological survey work from coal cores or by other methods (see above); calculations based on this gas emission are made of the individual air quantities needed to ventilate the developments, sections, seams, and the mine as a whole with allowances for:

—the air quantities needed for diluting the methane emitted from the wastes,

—the air quantities needed for ventilating various machine rooms, etc ;

—the air losses through ventilation structures and the wastes

The last method of calculation (for mines being planned) is more complicated than the generally accepted and earlier method based on coal output At present there is not yet enough information for comparing these two methods of calculation. Current Soviet Safety Regulations prescribe the calculation based on coal output

3. *Calculation of the air quantity on the basis of coal output* This method of calculation is prescribed by the Safety Regulations only for coal mines. It is based on the assumption that the quantity of

gas emitted in the mine is proportional to the coal output. This assumption is not completely correct, as the most recent research on gas liberation underground has shown; with the same output, the absolute quantity of gas released and the gas emission per ton depend on the speed of advance of the production face, the quantity of air passing along the face or, which amounts to the same thing, on the pressure of the air current, the condition of the roof rock (and, consequently, on the method of roof control) and the presence of nearby seams.

Depending on the gas category of the mine (CO_2 or CH_4) the method of calculation used is to choose the standard per ton of output (with a certain safety factor) so that the content of CO_2 or CH_4 at the maximum gas emission for the category of the mine shall not exceed the percentage fixed.

If q is the calculated standard quantity of air per ton of daily output, in m³/min, the total air quantity is obtained from the equation

$$Q = qT \tag{III-3}$$

in which T is the normal daily output of the mine, tons.

The calculated values of q for CH_4 and CO_2 are given in Table III-2

TABLE III-2 Air Requirements Based on the Emissions of CO_2 and CH_4

Gas category of the mine	Methane or carbon dioxide liberated per ton of daily output, m³	Required air flow, per ton of daily output, m³/min
Non-gassy	up to 5	1 0
1	up to 5	1.0
2	from 5 to 10	1.25
3	from 10 to 15	1 50
Above the categories	more than 15	Adequate to reduce the content of either CO_2 or CH_4 in the general return to 0.75% maximum

Note. In mines above the categories, the air quantity should not be less than 1 5 m³/min per ton of daily output of coal

When seams in a series being mined belong to different gas categories, the air quantity can be approximately calculated in the following way:

(1) all seams not higher than Category 3 are assumed to have the same category as the seam with the highest gas emission; this will allow a small excess of air;

(2) when a number of the seams are above the highest category, the air quantity is calculated according to Category 3 and distributed proportionally to the output; in this method some air deficiency may occur when a high proportion of the output is from seams which are above the worst category.

According to the Safety Regulations the most appropriate method is to make a calculation for each seam in accordance with its gas emission and in accordance with current levels of air required per ton of output; for seams which are above the worst category, the calculations should be made according to the percentage CH_4 content in the return airway; after this all the air quantities are added.

In the conditions of air supply of Table 3-2, the possible contents of CO_2 or CH_4 in the general return airway for the minimum and maximum emissions are indicated in Table III-3.

TABLE III-3 CO_2 and CH_4 Contents in the General Return Airway

Category of the mine	Carbon dioxide		Methane	
	gas emission, m^3/ton	content, per cent	gas emission, m^3/ton	content, per cent
1	1	0 07	5	0.35
2	6	0 33	10	0 56
3	11	0 51	15	0 70

Thus, in accordance with current standards of air quantity, the gas percentage (CO_2 or CH_4) in the return airway of a mine of Category 1 with a low gas emission is considerably lower than that permitted (1 and 0 75%); for a mine of Category 2 and in particular for one of Category 3, this safety factor is greatly lowered and finally in mines above the worst category it is completely nullified. Thus, included in these standard levels of air quantity there is an implicit safety factor which is fully adequate to ensure the necessary purity for the air in mines of Category 1 and even 2 with any fluctuations of the gas emission. But in mines of Category 3, particularly those with heavy gas emission, approaching the upper limit of the categories, during period of high gas liberation the gas content in the return airway must be expected to be above the permitted limits

Figure III-40 shows how, with changing gas emission in a mine and with the standard air quantities of Table III-2 being observed, the percentage gas content in the return airway of the mine changes, this change takes place jumpily and is in fact ill-founded; it is not clear, for example, why at a gas emission of 5 m³/ton the gas content allowed in the return airway is 0.36% but with a gas emission of 5.1 m³ per ton it may be only 0.28% and consequently the air

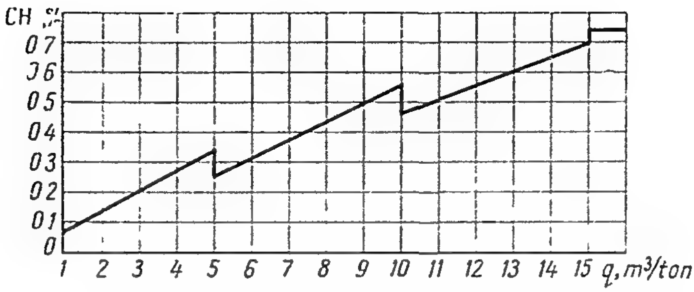

Fig III-40. Variation of the gas content in the general return airway with change in the gas emission of the mine

quantity must be increased by 25% for an increase of gas emission of only 0.1%. It is therefore preferable to calculate the air quantity according to Equation (III-2) and to multiply the result by some safety factor (see p. 528).

In planning the ventilation of metal mines it is usual to accept the standards of coal mines. Assuming that these standards give some safe excess of air, it appears possible to state for metal mines

Emission of CO_2 per ton of output, m³	m³/min/ton
up to 5	0.5
up to 10	0.75
up to 15	1.25

The calculation of the air quantity in metal mines by this method is made difficult by the absence of measurements of the CO_2 emitted in them. Usually they are arbitrarily related to Category 1 according to CO_2. In Krivoi Rog, the work carried out by the Dnepropetrovsk Mining Institute showed that of eight mines investigated, four belonged to Category 1, two to Category 2, and two were outside the categories for CO_2.

4. *Calculation of the air quantity based on the explosive consumption.* This method is used mainly in metal mines and in shale mines where the shale is so hard that it must be broken with explosives. The

calculation involves an assumption, approved by the Safety Regulations, that 1 kg of explosive produces 0.040 m³ of CO.

But since, according to the regulations, the poisonous products of the explosives must be diluted to a level not above 0.008% by volume, the air quantity which is needed according to the explosive consumption, will be

$$Q=\frac{Aa\cdot 100}{t\ 0.008}\ \mathrm{m^3/min} \qquad \text{(III-4)}$$

where A = amount of explosives fired at one time, kg

a = 0.040

t = time required for ventilation (re-entry period), minutes.

In fact this is not so. The equation would be reliable if the quantity of poisonous gases, Aa, formed by the explosion, were diluted in a closed space; then the addition to this space of 125 times the volume of air (1 0.008 = 125) would in fact dilute the poisonous gases down to 0.008%. In shotfiring at a face with an air flow, not only are the fumes continuously removed from the faces by the air flow (which is not accounted for in the equation) but in addition there is continuous dilution in the area of contact of the moving fume cloud with the intake air, which also is not considered in Equation (III-4).

The gases are therefore diluted more quickly than appears from Equation (III-4).

In Sections 5 and 6 of this chapter the most typical ventilation layouts of metal mines were described. It will be noticed that the space being ventilated only occasionally resembles the longwall face of a coal mine, mainly in mining thin veins; much more often the problem is to ventilate a stope and to determine its air requirement according to the explosive consumption.

From the point of view of ventilation the methods of mining ore deposits can be divided into the following main types:

(1) mining methods in which the face resembles a roadway ventilated by a through current of air;

(2) mining methods in which a large open stope has to be ventilated by a through current;

(3) mining methods in which the face is not ventilated and the air is supplied to the horizon of the scrapers and crushers; this includes most methods of bulk caving;

(4) mining methods in which the ore is mined at dead ends, e g. top slicing

The calculation of the air quantity for mining methods in which the face resembles a roadway. These methods include:

(a) methods of mining thin veins by an open stope, a filled stope, or a shrinkage stope;

(b) mining methods using top slicing with a through current of air into the stope.

The air quantity for these methods can be calculated according to one of the two equations:

(1) The equation of V.N. Voronin based on the fact that the explosive fumes are diluted by means of turbulent diffusion

$$Q = \frac{0.4}{t}\sqrt{ASL}\ \text{m}^3/\text{sec} \qquad \text{(III-5)}$$

where A = weight of explosives fired, kg
S = area of the cross section of the working space, m^2
L = length of the working space, m
t = duration of ventilation (re-entry period), minutes.

(2) The equation

$$Q = \frac{LS}{15t}\ \text{m}^3/\text{sec} \qquad \text{(III-6)}$$

In this equation the exchange coefficient includes a small safety factor equivalent to 4.

The quantity of air obtained should be checked for the maximum air velocity (not more than 4 m/sec) and for the minimum air velocity, preferably not less than 0.4-0.5 m/sec.

When some of the holes are fired during the shift, at the end of the shift when the remainder of the holes are fired, the air requirement is usually larger than during the shift and consequently the question arises whether there should be two ventilating conditions. But it is possible to accept this organisation of shotfiring without the need for two conditions Let t_1 be the duration of ventilation at the end of the shift, t_2 the duration of each ventilation period during the shift, and m the number of shots fired, then the amount of explosive which can be fired at the end of the shift, expressed as a fraction of the total consumption, will be obtained from the equation

$$n = 1 - \frac{mt_2}{mt_2 + 5t_1} \qquad \text{(III-7)}$$

For example, if $m=4$, $t_1=60$ minutes, and $t_2=10$ minutes, then

$$n = 1 - \frac{4\times 10}{4\times 10 + 5\times 60} = 0.88 \text{ or } 88 \text{ per cent}$$

Calculation of the air flow required for mining methods in which the working space is a large open stope ventilated by a through current. These mining methods include·

(1) open stopes with the ore broken by heading blasting, or by vertical holes the full depth of the horizon, or by sub-level stoping with long holes;

(2) sub-level caving, with a large mined-out stope above the draw raises;

(3) shrinkage stoping, the ore being broken at (or outside) the face.

A feature of all these methods of mining from the viewpoint of ventilation is the presence of a large chamber from which the blasting fumes pass out under the effect of a so-called free stream.

Considering the process of dilution of the poisonous fumes by the free stream of intake air passing along the stope or driven into it by ducting (the ventilation of a dead end of a development road considered in Chapter 15), V N. Voronin introduces the conception of "turbulent diffusion" which is the diffusion that takes place upon the interaction between the turbulent free stream of intake air and the mixture of air and fumes surrounding this stream in the space being ventilated.

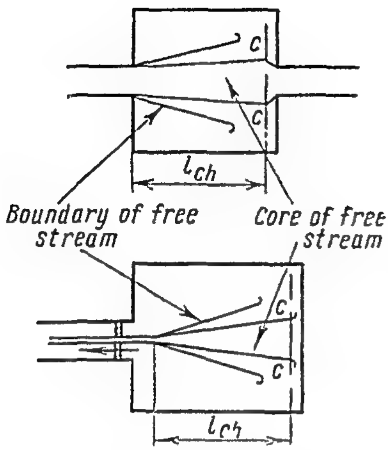

Fig III-41 Diagram of a free stream in a large open stope

Turbulent diffusion takes place hundreds or thousands of times more quickly than ordinary molecular diffusion. This difference is explained by the character of the two processes: instead of the slow exchange of the molecules of two neighbouring volumes of gas, turbulent diffusion takes place continuously with intensive mixing of the two gases The eddies in the free turbulent stream, by their cross movement, pass outside the limits of the stream; mixing with still air, and dragging it with them, they are somewhat retarded; on the other hand, when getting into the free stream the particles of the still air are carried away by it. As a result of these two processes, the total mass of the free stream increases and its boundary velocity diminishes, around the continuously eroding flow of intake air is formed a gradually increasing cloud of the more slowly moving mixtures of eddying intake air and foul air moving with it

The coefficient of turbulent diffusion, according to V N. Voronin, is the ratio of the mean concentration of gases c_m in the core of the constant mass *c-c* (Fig III-41) near the outlet from the chamber, to the mean gas concentration c_{ch} in the chamber. The greater the value of this coefficient, the more intense will be the dilution and removal of the gases out of the chamber The values of coefficients of turbulent diffusion are given in Appendix 6

The air quantity needed for ventilating these stopes is calculated from Voronin's formula

$$Q = 2.3 \frac{V_{st}}{kt} \log \frac{500A}{V_{st}} \text{ m}^3/\text{sec} \qquad \text{(III-8)}$$

where V_{st} = volume of the stope being ventilated, m^3
k = coefficient of turbulent diffusion
t = re-entry period, seconds
A = explosive charge fired, kg

Example. Calculate the air quantity required to ventilate a stope 20 m long, of cross section 40 m^2 if $S = 4\ m^2$, $a = 0.1$, $A = 1{,}000$ kg and $t = 30$ min

Solution. Since $\frac{al_{st}}{\sqrt{S}} = \frac{0.1 \times 20}{\sqrt{4}} = 1 > 0.38$ (Appendix 6), from the table we obtain by interpolation the value of the coefficient of turbulent diffusion, $k = 0.612$; substituting this value of k in Equation (III-8) we obtain.

$$Q = 2.3 \frac{800}{0.612 \times 30 \times 60} \log \frac{500 \times 10}{800} = 1.37\ m^3/sec = 82\ m^3/min$$

Methods of mining in which the working space is not ventilated by an effective through current of air These mining methods include.

(1) stoping with slicing methods of mining and long horizontal holes;

(2) sub-level caving (with a fan-shaped hole layout, and with long holes);

(3) methods of shrinkage stoping by horizons;

(4) block caving methods

The special features of these mining methods from the ventilation point of view are as follows:

(a) blasting, usually heavy charges of explosive are fired within an enclosure; this space can be limited by ore pillars at the sides and above, or there may be caved ground above and to one side; the floor of the stope consists of chutes connected by draw raises to the scraper drift or ore breaking drift;

(b) the fumes from the explosives are delivered partly through the draw raises to the scraper drift, partly (if the drilling drifts are blasted) into the raises, and partly expelled into neighbouring wastes and caved stopes; the rest of the fumes remain for some considerable time in the stope between the blocks of blasted ore;

(c) the main function of ventilation in these methods of mining is the dilution and removal from the mine of those fumes from the heavy blasting which are ejected from the stope into the airways nearby both downstream and upstream; secondly, the air supply ensures a positive air flow through the scraper drifts or crusher drifts to dilute and remove the fumes and the dust produced during ore drawing and breaking of the oversize blocks;

(d) since the air requirements during the period of intensified ventilation (after heavy blasting) and in the normal ventilating conditions (during ore production) are different, the necessary air flows for these two conditions will be different.

The peculiar processes of removal and dilution of the fumes after heavy blasting were first investigated by S I. Lugovsky. After further detailed investigation of these processes in the laboratory and in practice underground, he proposed and verified on a large scale a new method for calculating the air quantity required for heavy blasting.

Below several examples are given of his method of simplified calculation.

III-7 1 Ventilation of Large Stopes with a Slicing Method of Mining and Blasting by Horizontal Holes

The air quantity is calculated from the equation

$$Q = \frac{2.3V_a}{t} \log \frac{i A^2 b_a b}{V_a c_p (A b_a + V_{st})} \text{ m}^3/\text{sec} \qquad \text{(III-9)}$$

where A = actual quantity of explosive charge, kg

i = 0.175 when the ratio p of the volume of the stope to the charge is not less than 3 m³/kg; i = 0.250 for a ratio p between 3 and 10 m³/kg; i = 0.300 for p greater than 10 m³/kg

b_a = volume of all the gases formed when 1 kg of explosive is fired; with p_0 = 760 mm, and t = 0°C, it is taken as 0.9 m³/kg

b = 0 040 m³/kg, the quantity of poisonous gases produced by 1 kg of explosive

V_a = volume of the airways filled with gases upstream and downstream

t = re-entry period, minutes

c_p = final permitted concentration of gas, 0.00008 m³/m³

V_{st} = free volume in the stope, m³.

Example. Given A = 1,300 kg, the volume of the stope V_{st} = 9,100 m³, the volume of the return (downstream) airways = 8,500 m³, the re-entry period = 30 minutes Determine the necessary quantity of air for ventilation.

Solution. Preliminary calculations

(a) Let us find the volume of the stope per kg of explosive

$$p = \frac{9{,}100}{1{,}300} = 7 \text{ m}^3/\text{kg}$$

(b) For this value of p, i = 0 25

(c) The value of the arbitrary explosive charge producing a volume of gases equivalent to the sum of the gases from the ore produced in breaking ground and during crushing will be

$$A_{ar} = iA = 0\ 25 \times 1{,}300 = 325 \text{ kg}$$

(d) The total volume of the gases released into the working of the stope after the blasting will be·

$$C_t = A_{ar} b_a = 325 \times 0.9 = 293 \text{ m}^3$$

(e) The volume of the airways filled with gases will be

$$V_a = V_{ret} + C_t = 8{,}500 + 293 = 8{,}793 \text{ m}^3$$

Substituting these values in Equation (III-9), we obtain

$$Q = 2.3 \frac{V_a}{t60} \log \frac{tA^2 b_a b}{V_a c_p (A b_a + V_{st})} =$$

$$= 2\ 3 \frac{8{,}793}{30 \times 60} \log \frac{0\ 25 \times 1{,}300^2 \times 0\ 9 \times 0\ 04}{8{,}793 \times 0\ 00008\,(1{,}300 \times 0\ 9 + 9{,}100)} = 3\ 6 \text{ m}^3/\text{sec}.$$

This quantity of air represents the conditions of intensified ventilation after heavy blasting When the poisonous gases have been reduced to the permitted concentration after a time t, the quantity of air flow can be reduced to the "normal" level, ensuring dilution of the gases released from the ore being drawn and caused by the breaking of the oversize lumps, in addition, this quantity of air should remove the dust from the scraper drift or breaker drift

Bearing in mind that the concentration of carbon monoxide is approximately proportional to the quantity of ore produced, S I Lugovsky proposes, for calculating the "normal" air flow, the following equation

$$Q_n = \frac{40\ 3}{t'} \sqrt{A_{ar} V_d} \text{ m}^3/\text{sec} \qquad \text{(III-10)}$$

where t' = re-entry period after blasting during ore breaking; this can be assumed to be 5 minutes (300 seconds)

A_{ar} = arbitrary charge of explosive with the amount of gases released equal to the sum of the gases emitted from the ore drawn and that being broken

V_d = volume of the drift being ventilated calculating from its beginning to its junction with the gathering airway, m³

The value of the arbitrary charge

$$A_{ar} = A_1 + A_2$$

where A_1 = quantity of explosive from the broken ore

A_2 = volume of explosive used in breaking

To determine A_1, S I. Lugovsky gives the equation:

$$A_1 = \varepsilon \frac{P_d f t'}{\gamma t_{dr} b_a} \qquad \text{(III-10a)}$$

in which ε = coefficient which takes account of the intensified release of gases in the early period of drawing; from practical experience it has been found to be 2.7

P_d — daily quantity of ore taken from the draw points, tons

f — bulking factor for broken ore (m^3/m^3) which can be taken equal to 0.3

γ — specific weight of broken ore, t/m^3

t_{dr} — daily duration of ore drawing, sec

b_a — total emission of gases from 1 kg of explosive, equal to 0.9 m^3/kg

t' — re-entry period, assumed equal to 5 minutes

Example. Given: the number of places from which ore is drawn = 2; V_r = 120 m^3; P_d in the course of 3 shifts is equal to 115 tons; [illegible]; γ = 2.5 tons per m^3; t_{dr} = 20 hours.

Solution:

$$I_1 = \frac{P_d f t'}{\gamma t_{dr} b_a} = 2 \cdot \frac{[illegible] \cdot 115 \cdot 0.3 \cdot 5 \cdot 60}{2.5 \cdot 20 \cdot [illegible] \cdot 0.9} = 0.52 \text{ kg}$$

$$I_{ar} = I_1 + 1 = 0.52 + 1 = 1.52 \text{ kg}$$

$$Q = 2\frac{30.3}{t'}\sqrt{I_{ar}V_r} = 2 \cdot \frac{30.3}{[illegible]}\sqrt{1.52 \cdot 120} = [illegible] \text{ m}^3/\text{sec}$$

Checking on the removal of dust: suppose the section of the drift is 5 m^2 and the minimum air velocity which will remove the dust is [illegible] m/sec; then $Q_{dust} = 2 \cdot 5 \cdot 0.5 = 5$ m^3/sec, i.e. more than that required for diluting the gases.

III-7.2 Ventilation of Stopes in the Method of Block Caving

The air quantity in block caving is calculated from Equation (III-10)

$$Q = \frac{30.3}{t'}\sqrt{I_{ar}V_r} \text{ m}^3/\text{sec}$$

in which, depending on the type of blasting, we assume that:

(a) when the mass is undercut, the re-entry period t is 30 minutes, including in the gas-filled volume all the airways up to the surface.

(b) in the drawing of ore and the breaking of the oversize, the time of ventilation is 5 minutes and the gas-filled volume is only that of the scraper drift from the hoist to the junction with the gathering airway.

The calculated air quantity is checked for removal of airborne dust (see page 521).

III-7.3 Ventilation of Stopes in the Method of Induced Block Caving

S.I. Lugovsky gives the following equation:

$$Q = \frac{50}{t}\sqrt{I_{ar}V_a} \qquad \text{(III-11)}$$

where A_{ar} = arbitrary charge equal to iA; the value of i being assumed equal to 0.157 for stopes with one upper contact surface; 0 125 for stopes with one upper and one side surface; 0 115 for stopes with one upper and 2 or 3 side surfaces;

V_a = volume of gas-filled airways equal to

$$V_{ret} + C_t$$

in which V_{ret} = volume of gases on the return air side and C_t = total gas emission equal to $A_{ar}b_a$.

III-7 4 Ventilation of Pyrite Mines

The gases produced in blasting in pyrite mines include sulphur dioxide, pyrite ores tend to spontaneous combustion Owing to these two features, the following should be borne in mind in ventilating pyrite mines·

1. Since sulphur dioxide is 2 2 times as heavy as air, it tends to accumulate at the floor of the stope, thus prolonging the re-entry period, it has been proposed to blow compressed air into accumulations of sulphur dioxide (using the mine rescue force) from the lower sub-level, providing powerful mixing of the gases with the air and rapid removal of the gases from the stope; the method has been successfully used in copper pyrite mines in the Urals.

2. Self-heating of pyrite ore results in an increased temperature of the air in so-called "heated" sections; the temperature reaches or even exceeds the permitted level (26°C). To cool the air, the following measures are recommended, apart from firefighting preventive action

(a) adequate supply of air to the working places by installing auxiliary fans in hot stopes, working usually through a stopping, but in certain cases without a stopping, the type described in Chapter 12 as pusher fans;

(b) reduction of the length of the air course through airways with heated walls, and elimination of the recirculation of air,

(c) strenuous efforts to reduce air leakages at the stope;

(d) the ventilation ducting should be brought as close as possible to the face, ordinarily 4 or 5 m, but not more than 6.5 m (to avoid damage to metal ducting, the last 3-5 m length should be made of rubberized canvas),

(e) fans should be of blower type.

The hot places should be located by temperature surveys

III-7 5 Ventilation when Undercut Pillars of Ore are Blasted

Depending on the thickness and type of overburden, the fumes from explosives can, to some degree, penetrate to the surface (thus facilitating the subsequent ventilation) or be sucked into mine workings G V. Malakhov considers that after blasting the fumes penetrate into caved ground for 40 to 60 m. Depending on the method of ventilation the fumes can gradually be sucked out of the caved stope (with exhaust ventilation) or be driven into caved ground by forcing ventilation.

The air quantity can be calculated by Equation (III-12).

The arbitrary charge A_{ar} is equal to iA, where A is the actual explosive charge, kg; on the basis of observations at the metal mines of Krivoi Rog, Lugovsky states that "for voids under drift deposits or separated from them by one or two horizons, i should be equal to 0 095; and for voids under several worked-out horizons, it should be 0 124".

The volume V_a can be taken to be equal to the sum of the airway volumes from the point of the blast to the outlet of the air at the surface plus the explosive fumes

$$V_a = V_{ret} + A_{ar} b \qquad \text{(III-12)}$$

where b is the total gas emission of 1 kg of explosive (for ammonites 0 9 m^3/kg)

A comparison between the calculated and actual durations of ventilation showed good agreement

Before heavy blasting, the following precautions should be taken.

(a) all fans should be stopped,

(b) doors should be opened in all stoppings;

(c) the districts where blasting is taking place should be fenced off by gridded stoppings (e g made of sleepers) to deaden the explosive waves,

(d) the necessary action for intensifying the ventilation of the section should be taken after blasting.

In copper pyrite deposits, to avoid explosions of sulphide dusts (see Part One) the following additional recommendations are made

(e) using fire hoses, the dust should be washed from the walls of neighbouring airways, as well as from broken ore, and so far as possible from the walls of stopes;

(f) neighbouring airways should be supplied with watering equipment, and the water should be turned on two hours before the blast;

(g) adequate water should be supplied to the section involved.

III-7 6 Calculation of the Quantity of Air Required to Remove Dust

Soviet Safety Regulations prescribe that air shall be provided to remove dust, but do not indicate how the quantity shall be calculated. Although in recent years a large amount of investigation on the dustiness of mines has been undertaken, there is no generally accepted method of calculating the amount of air required to remove dust.

The ventilation of dusty workings is a necessary part of dust suppression. But, as research has shown, it can only be effective in certain conditions. If these conditions are not observed, ventilation can change from a help, reducing the dustiness of the air, to a hindrance, increasing dustiness

This is explained by the fact that higher air velocities begin to raise the dust which formerly settled on the walls and roof of the workings

Consequently the quantity of air calculated to reduce the airborne dust to the permitted level, as for gas, should be checked for air velocity

As far as gases are concerned, the check for velocity is simple because it is limited by the condition of turbulence of the air current, for which the limit underground is generally some 10-20 cm/sec.

The establishment of effective standards for airborne dust, and so much the more for the optimum air velocity is made difficult by the fact that they are associated with so many factors (the intensity of dust formation, the size and specific gravity of the dust, etc.), and in particular whether water spraying is practised.

Consequently the required minimum air velocity varies with the conditions, from a few tenths of a metre to well above 1 m/sec, and generally speaking can only be reliably established by preliminary tests in each mine. Thus, for example, numerous observations in the mines of Kazakhstan, Krivoi Rog, and the Urals have shown that in the driving of horizontal roadways, ventilation is only effective at air velocities which vary with the dust formation from 0.2 up to 0.7-0.9 m/sec. For workings shaped like a coal face, the greatest reduction of dustiness according to the Moscow Mining Institute is found at air velocities of about 2 m/sec. When the air velocity rises to 4 m/sec, the dustiness rapidly increases because the dust is lifted from the surfaces of the walls and roof at the face

It must be remembered that in very dusty conditions when ventilation is ineffective, the dust will have to be suppressed by a complex of measures, in particular by plentiful watering of the surface of the walls and roof of the workings and the broken mineral as well as by injection of water under pressure into the face.

TABLE III-4 Equations for Calculating Ventilating Air Quantities

Operation	Ventilation by intake air		Ventilation with slightly dusty air	
	Air quantity needed for reducing the dust content of the air in the mixing zone or that breathed by the driller, m³/sec	Air quantity needed for reducing the dust content throughout the length (volume) of the airway, m³/sec	Air quantity needed for reducing the dust content of the air in the mixing zone or that breathed by the driller, m³/sec	Air quantity needed for reducing the dust content throughout the length (volume) of the airway, m³/sec
Blasting	$Q=\frac{V}{k\ t}\log_e\frac{n_0}{n}$	$Q=\frac{S}{t}\sqrt[3]{\frac{L^2 l_2}{P_1^2}\frac{n_0}{n}}$	$Q=\frac{V}{k\ t}\log_e\frac{n_0-n_0'}{n-n_0'}$	$Q=\frac{S}{t}\sqrt[3]{\frac{L^2 l_2}{P_1^2}\frac{n_0-n_0'}{n-n_0'}}$
Drilling	$Q=\frac{b'}{k\ n+0\ 61\ \frac{Ab'}{adl_2}}$	$Q=\frac{b'}{n}$	$Q=\frac{b'}{k\ (n-n_0')+0\ 61\ \frac{Ab'}{adl_2}}$	$Q=\frac{b'}{n-n_0'}$

Notation used in Table III-4

Symbol	Ventilation planning of drivages	Ventilation planning of production faces (stopes)
V	Volume of the mixing zone, m^3	Volume of the stope, m^3
S	Cross-sectional area of the airway, m^2	Ditto
L	Ventilated length of airway, m	Length of stope, m
P_1	Leakage factor	Ditto
l_2	Distance from the face to the end of the ventilation ducting, m	Ditto
k	Coefficient of turbulent diffusion	Ditto
n_0	Original dust content in the mixing zone, mg/m^3	Original dust content in the stope, mg/m^3
n_0'	Original dust content of the intake air, mg/m^3	Ditto
A	A factor depending on the number of rock drills at work, and equal to 0 8 for one, and to 0.5 for two rock drills	Ditto
b'	Rate of dust formation, mg/sec	Ditto
d	Diameter of ducting, m	Ditto

Note To determine the total air flow through the mine, all the flows, obtained for the individual work places from the equations in the table, should be added together

The air quantity needed for the ventilation of a dusty working can be calculated by the formulas in Table III-4 which are based on theory and on tests.

III-7.7 Methods of Calculating Air Flow for Dust Conditions

Since the points of dust formation are generally the production or development faces, where the dust is formed mainly in drilling and loading, the air flow for airborne dust in the mine as a whole can be calculated by adding the air quantities needed for ventilating the individual faces (Part Two, Chapter 15).

One of the methods of calculating the air flow was proposed by V.N. Voronin. This method involves the assumption that the air flow "for dust" should evidently be calculated for the dust formed during the operations, including drilling and loading of mineral.

All metal mines, depending on their dustiness, can be divided into the following categories: Category 1 with dust smaller than 5 to 10 microns, in quantities up to 1,000 mg/ton; Category 2 from 1,000 to 5,000 mg/ton; Category 3 from 5,000 to 20,000 mg/ton and mines above the categories more than 20,000 mg/ton

Category 1 includes mines with wet drilling and water spraying using natural block caving and similar mining methods.

Category 2 includes mines with wet drilling and water spraying using induced block caving or sub-level caving with shrinkage stoping or filled stopes.

Category 3 includes mines with top slicing and sub-level caving.

Mines above the categories are those where no dust suppression measures are used.

The following tentative standards are proposed for the air supplied per ton of daily output provided that the mineral is loaded on every shift, and that the dust concentration in the intake air is equal to 0 2 mg/m^3

(1) metal mines of dust Category 1. up to 0 4 m^3/ton per min
(2) metal mines of dust Category 2 from 0 4 to 2 m^3/ton per min
(3) metal mines of dust Category 3 .. from 2 to 8 m^3/ton per min
(4) metal mines above the categories more than 8 m^3/ton per min

Since it is difficult to supply air quantities exceeding 2 m^3/ton of output, the general dust suppression condition for the air flow in mines of Category 3 and higher is impossible and therefore mines should be reduced to a lower dust category by the use of the widest dust suppression measures

The proposals of V N. Voronin deserve attention, they should be checked in practice

A simplified method of calculating the air flow has been proposed by V M Kiselev and S. E Ovcharenko It is based on an air velocity which ensures active removal of the dust from the working places. 0 5-0 8 m/sec in the stope and 0.25 m/sec in the sub-levels; for mines of varying output working ore with a Protodyakonov hardness (M M Protodyakonov's scale) of up to 10, Fig III-42 is proposed. It was drawn up for the conditions of Krivoi Rog and its various mining methods including sub-level caving, block caving, and sub-level stoping; with harder ores a correction factor is introduced, which is 1 25 for ores of a Protodyakonov hardness of 18 Curve *1* refers to work in three shifts of 8 hours with 307 working days per year;

curve 2 refers to two-shift working for 8 hours each (307 days): curve 3 is for two-shift working with 255 working days per year.

Check on the maximum and minimum air velocities. According to the Safety Regulations the air velocity in the faces should not be less than 0.25 m/sec: the minimum air velocity in developments was discussed in Chapter 15.

The maximum air velocity should not exceed:

(a) in the faces and development roads. 4 m/sec;

(b) in the crosscuts. main haulage roads and return airways, 8 m/sec;

(c) in other airways, 6 m/sec:

(d) in shafts equipped for winding men and mineral, 8 m/sec; in shafts equipped for winding mineral alone, 12 m/sec; and in shafts not equipped for hoisting or in fan drifts, 15 m/sec.

The maximum velocity is limited because high air velocities tend to give men chills and to raise the dust in the airways; the latter is important in connection with the increasing mechanization of mining which also forms dust: in many countries in this connection there is an effort to reduce the air velocity in faces to 2-3 m/sec.

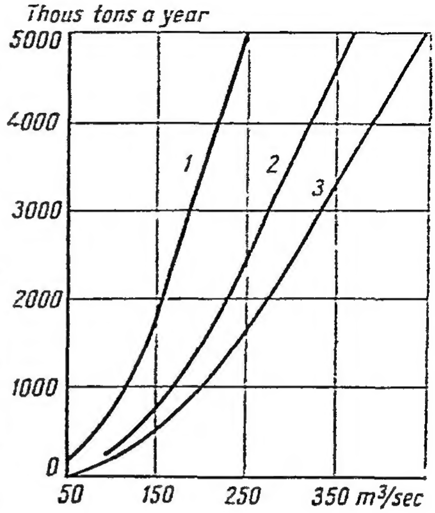

Fig. III-42. Curves for determining the total quantity of air required by the mine in relation to mine output and shift organization

The air velocity prescribed by the Safety Regulations provides for effective ventilation of mine workings; at lower velocities the air stagnates in the irregularities of the timbering. The following points should also be mentioned in connection with air velocity:

(a) with descensional ventilation in gassy coal mines the air velocity should be at least 0.8 to 1 m/sec;

(b) in metal mines at dusty places and where blasting takes place during the shift (breaking the oversize), e.g. in the scraper drifts, the air velocity should be not less than 0.6 to 0.8 m/sec;

(c) in ventilating deposits worked by large open stopes. e.g. rock salt mines, it is generally impossible to provide a minimum air velocity of 0.15 m/sec from the main mine fan: in this case small stope fans should be used, which provide energetic movement of the air and remove the dangerous fumes and dust. Sometimes compressed air is released, which. however, is uneconomical.

Determining the length of the face, based on the ventilation condition. Limitation of the air velocity in the face to 4 m/sec creates no difficulty in the planning of mines of methane Categories 1. 2

and 3, however, for mines above the categories, beginning with gas emissions of about 35 m^3/ton, the face length, to satisfy the ventilation conditions, may have to be shortened in accordance with the rational use of cutter-loaders

To determine the face length on the basis of ventilation the following equation is used

$$L = \frac{60 v b m_1 \varphi}{q l m_2 \gamma c} \text{ m} \tag{III-13}$$

where L = face length, m
v = limiting permissible air velocity, m/sec
b = width of the working space in the face, m
m_1 = total thickness of the seam, m
m_2 = workable thickness of the seam, m
φ = coefficient of restriction of the air flow, allowing for the blocking up of the face
γ = specific weight of the coal, tons/m^3
c = coefficient of extraction of the coal
l = daily advance of the face, m
q = standard of air quantity equal to $\frac{100\, q_{gas}}{24 \times 60} = 0.07 q_{gas}$, m^3/min for each ton of daily output, where q_{gas} is the methane emission from the section, m^3/ton.

If we assume that $m_1 = m_2$; $\varphi = c$ and $v = 4$ m/sec, $\gamma = 1.3$ and the width of the working space is equal to twice the daily advance, then substituting these values in Equation (III-13) we obtain

$$L = \frac{5{,}270}{q_{gas}} \tag{III-14}$$

Equation (III-14) would be reliable if, first of all, all the gas emitted in the section passed through the face, and secondly if all the air passed through the face But this is not so in reality. Part of the gas emitted in the section usually does not pass through the face but avoids it, passing through the waste, similarly part of the air avoids the face by passing through the waste and the width of the working space (b) in Equation (III-13) should be increased by the factor k, stated in the table below

Method of roof control	Immediate roof	Value of the factor, k
Full caving	Sandstone	1 30
Full caving	Sandy shale	1 25
Full caving	Clay shale	1 20
Smooth lowering	Clay shale	1 15
Partial packing	Clay shale	1 10
Full packing	Clay shale	1 05

Unfortunately we do not yet possess enough basic data to enable us in planning the ventilation of a mine to predetermine the gas emission at the return airway, in seams which are not close to other seams, we can take $q_{mine} = q_{section}$; in seams close to unworked seams, the percentage content of gas at the return airway must be obtained from the results of gas surveys in neighbouring working mines.

We must point out that Equation (III-14) is based on the condition that the gas emission in the face is directly proportional to its length; however, there are indications that after a certain length of face has been reached, the absolute quantity of gas released does not increase, similarly it is known that in some faces, increase of the face length reduces the gas emission per ton; finally the speed of advance of the face also affects the gas emission per ton and reduces it All these variables and practical experience [in some mines there are faces with lengths exceeding that given by Equation (III-14)] force us to consider Equation (III-14) as approximate, not including all the facts, and therefore requiring correction, for individual seams the corrections can be found from underground observations; to give a general correction is not now possible since there is not enough knowledge.

Safety factor for air quantity. The quantity of air obtained from calculation should be multiplied by a safety factor to take account of underground leakages, the ventilation of reserve faces or stopes and the individual ventilation of rooms; in addition, it should cover the air requirement caused by inequality in the gas emission and in the output if it is also uneven. In the publication "Basic Statements for Planning a Complex Reconstruction Project and Modernizing Mines in the Donets Basin" it is taken as 1 45; for metal mines there is no established safety factor.

Current standard limits contain implicitly a certain safety factor which expressed as such may vary from 2 1 to 10 7 for mines of methane Category 1; for mines of methane Category 2 it varies from 1.3 to 2 3 and for mines of Category 3 it varies from 1 07 to 1.5 Bearing this in mind and also that with air leakages through the ventilation structures and wastes, of the magnitudes indicated in Chapter 14, the total air losses in the mine should not reach 35 to 45% of the air quantity reaching the face, it is therefore possible to establish different air safety factors for mines of different categories. Another solution of the problem of safety factors is the following: at first the intensity, locations, and variations of the various sources of gas are found out, and then considering that part of the air leakage is useful (see p 373) the safety factor can be calculated. This method, proposed by the Department of Ventilation of the Moscow Mining Institute, requires a preliminary gas survey of the mine.